OCTOBER 1952

MINING ENGINEERING

FREEZING FOR SHAFT SINKING

MINING ENGINEERING

Incorporating Mining and Metallurgy, Mining Technology and Coal Technology

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FEATURES

ARTICLES

TRANSACTIONS

— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Project Manager or Construction Engineer, 48, married. Twenty-five years' experience on mine, mill and smelter construction work, design and operation, U. S. and Mexico. Speak Spanish. Now employed; available 30 days notice. Prefer U. S. or Mexico. M-698-489-E-1-San Francisco.

Mining Geologist, 53, single. Eleven years' intensive field experience giving a completely practical competence for the smallest or largest of areas; plus the highest possible grade of academic background and independent research qualifying for top rating. Available in thirty days or less. Any location acceptable. M-699-3611-E-7-San Francisco.

POSITIONS OPEN

Exploration Geologist, under 40, with at least five years' experience prospecting, examination, mapping and drilling experience for metal mining fields. Must speak Spanish. Salary, $5000 to $7000 a year. Location, Cuba. Y-7510.

Mining Engineer with broad exploration experience, young, single status, and must be in excellent physical condition, as nature of work will be under very primitive conditions. Salary about 25% more than applicants last salary rate plus travel expenses and living allowance. Duration until July 1953. Location, Burma. Y-7498.

Assistant Mine Superintendent, not over 40, for iron ore mining operation; open stope; should know rock as work is done by upraising. Salary, $6000 a year to start. Location, eastern United States. Y-7484.

Ore Geologist or Mining Engineer with experience in surveying primary and alluvial deposits to take charge of an expedition making a survey of tin ore deposits. Salary open. Location, South America. Y-7231.

Mill Superintendent to take charge of a 100-ton flotation gold mill. Must speak Spanish. Elevation, 9000 ft. Salary open. Location, Argentina. Y-7216.

Chief Engineer, mining graduate preferred, but will consider civil or mechanical, with about ten years' experience, to include either construction or underground operations. Mining is open-pit but transportation problem is involved so should have some experience in mechanized operation. Transportation paid; housing available; salary open. Location, West Indies. Y-7180.

Project Mining Engineer, with at least ten years' experience in design, layout and cost estimating for new mines and production plants. Should be capable of supervising construction on all such projects. Must have rather broad practical and technical knowledge of mining and milling operations. Salary, $6000 to $6600 a year. Occasional travel. Location, Texas. Y-6976.

Engineers. (a) Assistant Chief Engineer, with underground experience, and preferably a working knowledge of Spanish. Salary, $4200 a year plus bonus. (b) Mill Superintendent with broad experience in milling. Should have working knowledge of Spanish. Salary, $5400 a year plus bonus. (c) Mine Foreman, with general mining experience, particularly shrinkage and cut and fill methods of mining. Salary, $5400 a year plus bonus, under standard three-year contract. Location, Bolivia. Y-6402.

Mill Superintendent, 35 to 40, with experience in base metal, preferably copper. Must have had concentrator experience. Two to three years' contract. Salary open. Location, Northern Rhodesia. Y-5903.

Engineers. (a) Metallurgist with minimum of five to ten years' experience in a mill for metal mining company. (b) Assistant Mining Engineers, for metal mining operations in the Philippine Islands. Salaries open. Y-5022.

Chief Engineer, up to 45. Training and experience in mining, chemical or metallurgical engineering. To administer and coordinate engineering for chemical operating company handling design and construction of new facilities. Salary up to $20,000 a year. Employer may negotiate fee. Location, Chicago, Ill. R-9152.

Plant Engineer, up to 55. With at least three years' experience as plant engineer or maintenance engineer in mining plant. Should have knowledge of surface mining and process plant equipment. Duties will involve plant engineering and maintenance work for surface mining and processing operation of nonmetallic products, for a manufacturer of asbestos. Salary, $8500 to $10,000 a year. Employer will negotiate fee. Location, California. R-9127.

Plant Engineer, mechanical or electrical, 30 to 50, to supervise mechanical plant and equipment for mining operation including electrical stacker dredges, hydraulic and mechanical stripping operations equipment, 10,000 K.W. steam power plant, mechanical shops, trucks and bulldozers. Salary, to $12,000 a year depending upon experience. Location, Alaska. T-9172.

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One million tons of rock make a pile 264' high. Thousands of Eimco loaders now in operation will easily load this amount of rock per hour, every hour.

Calculated on the basis of a right pyramid with square base and equilateral triangular sides, rock at 165 lbs. per cu. ft.

MOST TUNNEL DRIVERS USE EIMCOS:

Shown below are pictures of Eimco equipment used in driving five famous tunnels. Eimcos have been used on scores of other tunnels.

MOST HARD ROCK MINERS USE EIMCOS:

Shown below are pictures of Eimco loaders used in several mines. Hundreds of others are being used in mining properties throughout the world.

Eimco loaders are preferred equipment wherever rock is loaded underground. The Eimco quality of material and manufacture assures the user of dependability on the job. Greater economies in operating costs and time are available only with Eimco Loaders. Write for information on the Eimco to fit your job.

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A393

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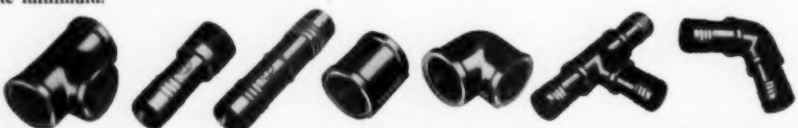
"Carlon costs less!"

"We've really saved on piping costs since we started using CARLON. It's much lighter and easier to handle than heavy metal pipe . . . requires fewer fittings . . . makes-up faster . . . and lasts many times longer with no maintenance. What's more, we don't pay a premium price for these money-saving features!"

CARLON plastic pipe gives you superior quality at low cost. For your protection, every foot is factory-tested for more than 8 hours at higher-than-working pressures. In addition, it is guaranteed against rot, rust and electrolytic corrosion and won't accumulate scale or sediment.

CARLON — the first real pipe that is plastic — has proven to be the answer to mine piping problems because it cuts installation and maintenance costs to absolute minimum.

NOM. SIZE	O.D.	I.D.	EST. BURST P.S.I.	WT. LBS. PER FT.	SHPG. LENGTHS
½"	0.840	0.622	540	0.103	400 ft. coils
¾"	1.050	0.824	350	0.140	400 ft. coils
1"	1.310	1.070	200	0.181	300 ft. coils
1¼"	1.660	1.380	200	0.267	300 ft. coils
1½"	1.900	1.610	200	0.320	250 ft. coils
2"	2.378	2.070	170	0.445	200 ft. coils
2½"	2.875	2.469	170	0.680	200 ft. coils
3"	3.504	3.070	165	0.910	100 ft. coils
4"	4.504	4.030	150	1.250	25 ft. str.
6"	6.630	6.070	115	2.230	25 ft. str.



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Letters to The Editor

One-Third—Not One-Half

I am much pleased by your presentation of my article and flattered to the extent that the cover design refers to it.

Incidentally, as between folks who like to work with words, refer to line 6, second full paragraph, page 773 (August *Mining Engineering*): Would "for" preceding technological have changed the percentages in lines two and three of the bold-face heading on page 770? The intent was coequality among three, *(studies of mankind, of science, and of technological empirics)*.

Thirty-three pct for "man" studies is a pretty big bite for engineering schoolmen; 50 pct they won't even examine, let alone nibble at.

Arthur F. Taggart

For those of you without a copy of the August issue handy, the sentence referred to read: The plea herein is to make the overall time allotment for studies of mankind coequal with those for science and technological empirics. . . . Professor Taggart suggests that insertion of the word "for" would clarify the intent of coequality among three divisions of study: The plea herein is . . . for studies of mankind coequal with those for science and (for) technological empirics. Editor.

Another Viewpoint on Foreign Mining

As a citizen and mining engineer from Chile the July issue of *Mining Engineering* on the problem of Foreign Mining gives me opportunity to refer to some statements I have heard in this country. These statements, in my opinion, do not reflect accurately the situation existing in foreign countries, especially in South America.

The U. S. depends to a large extent upon raw materials from abroad, and South America is probably the largest and nearest supplier. At the same time it is the nearest foreign market for American industry. But how can South America buy from the U. S. without dollars?

Nationalism and communism are charged when foreign governments declare mines a national heritage, restrict remittance of profits, require a certain percentage of native personnel, if they (as Mr. Weiss says) regard mining as a "catch-all" for raising revenue.

Chile produces 15 times more copper per capita than the U. S. and lacks many sources of revenue which this country has. Can you blame her for stressing the national importance of mineral wealth?

Comparing the price increase of articles which Chile imports, mainly from the U. S., with copper price from 1938 to date, we find that the latter would have to be close to 37¢ per lb, or 1.5¢ higher than the price Chile is receiving now. Only the need for survival causes these countries to ask certain prices for their minerals. Due to differences in price between raw materials and manufactured products money is actually flowing toward the U. S. and away from producing countries.

Concerning export of profits, if a nation's principal source of wealth is in ore deposits and profits do not return, it cannot industrialize and is supported only by export of raw materials. There is no stimulus for youth to engage in research or in technical professions. With economic instability produced by world market fluctuations comes political instability.

An average Peruvian miner receives a salary of 50¢ to $1 per shift. A Chilean worker might receive as much as $3 per day. With ample margin for lower efficiency the cost of labor is still much cheaper than in the dollar or sterling areas, and this

(Continued on page 913)

difference is usually compensated by extra taxes or special rates of exchange.

Recognizing all that American companies, the government or foundations have done for the people in general it is irritating to a South American to speak about Foreign Aid or Give-Away policy. No humanitarian philosophy, but cold and calculated money making on the side of the U. S. was the basis for the copper agreement with Chile in May, 1951, and in the negotiations with Bolivia.

Consider simultaneously the problems of popularity of the companies and the lack of sufficient well-trained technical men. How can you expect a South American college graduate with 6 years study to go to work for a company where he gets paid $100 to $150 per month and lives in type B houses, does not belong to the Staff Club, whereas his American colleague with 4 years training receives $300 per month, lives in type A houses, etc., without feeling a foreigner in his own country?

After nearly a year in this country and having learned that the people in the U. S. are different, and better, than they are believed to be abroad, my wishes are for a broad "mineral policy of cooperation" between the U. S. and South America, extended to all countries. But this requires better understanding—not an attitude of contempt—and a more human policy—instead of strictly money making negotiations on the side of the U. S.

These countries must be given a chance to develop their industries, primarily the fabrication of products from their ores. A higher dollar return from manufactured products and reinvested capital would improve the foreign trade and revenue situation, standard of living would rise, and danger of political instability and extreme movements would disappear.

I dare to believe that most of the countries of the free world would cooperate enthusiastically with such a policy. As things are now, many of them might follow the example given by Iran and nationalize their mines and other industries operated by foreign companies, with 100 pct of their population decided to support the greatest hardships in the fight against economic imperialism.

Mr. Werner Joseph
State College, Pa.

See Trends, page 938, Editor

Public Relations and Mining

An impressive amount of propaganda concerning the mining industry has been reaching the public for some time. Everyone is getting into the act. Each has his idea of what should be done to and for the industry.

The easily available information on mining for the general reading public is an occasional item concerning a rich find, a mine accident, statements in time of labor strife, or views of those people with an axe to grind. The reading is spectacular, but leaves a biased and unfavorable picture of the industry in the minds of the readers.

Our technical magazines are full of good material on all phases of the industry and its problems, but it reaches only a very few, speaking in terms of public opinion.

This is one field in which each of us from the boss down has an opportunity to do a good turn for the industry. We can put a little effort toward presenting our case to the public in articles written for public consumption. This will present our side of the picture and inform the man on the street that there is more to mining than shovelling gold out of an underground vault.

I believe we have a good case. Let's get it to the public before it's too late, and in a form that is both easy to read and easily accessible—the popular magazines.

E. O. Bracken
Kellogg, Idaho

MEET THE AUTHORS

Alan Stanley *(Titanium Dioxide Analysis of MacIntyre Ore by Specific Gravity, P. 971)* is currently head of the General Metallurgical Group of the National Lead Co. He has been with National Lead since 1946, after serving with the USAAF as a first lieutenant of engineers during World War II.

A native New Yorker, he graduated from Virginia Polytechnic Institute in 1942 with a Bachelor of Science in Chemical Engineering. Since joining National Lead he has been a flotation supervisor, in the high tension pilot plant, and done sinter testing. He is an AIME member and presented one other paper before the Institute, *Sintering Characteristics of Minus 65 and 20 Mesh Magnetite*. Stanley plays the piano in his leisure moments.

Cleland N. Conwell *(Seismic Survey for Bedrock Depth Determination, P. 954)* has his own geophysical company and is currently working for the Seattle, Wash., Department of Lighting. The power company in Seattle is city-owned. Several hydroelectric plants are in operation and plans are underway for additional power dams on the Skagit River. Conwell is making a

CLELAND N. CONWELL

seismic survey of the proposed dam sites. He is an enthusiastic fisherman when not out in the field on business, and also numbers home buildings and photography among his active interests.

Conwell holds a Bachelor of Arts in Geology from the University of Colorado and a diploma in structural engineering from the United States Armed Services Institute. This is his first paper for the AIME.

John Griffen *(The Tromp Heavy Media Process, P. 967)* follows hobbies which include gardening, golf, and stamp collecting when not busy at his job as a sales and consulting engineer with the McNally Pittsburgh Mfg. Corp. He has presented many papers before the AIME.

JOHN GRIFFEN

After graduating from Lehigh University with a degree in chemical engineering, and a key from Tau Beta Pi, Griffen was employed by the Lehigh Coal & Navigation Co., in its coal briquetting plant, as assistant fuel engineer. He then became assistant to the president of Harrison Bros. & Co. By 1915, Griffen was with the Hudson Coal Co. as a fuel engineer.

Henry Schwellenbach *(Eliminating Hand Picking at the Mt. Hope Mine, P. 950)* has presented one other paper before the AIME, *Effect of Reducing Rod Mill Feed Size*. He was a miner in California and Nevada gold properties before going to work for Anaconda in Montana. Later, he was plant foreman with Andes Copper Co. in Chile, and a mill superintendent in Nevada. He is presently mill superintendent for Warren Foundry & Pipe Corp., Mt. Hope, N. J.

What's ahead for you?

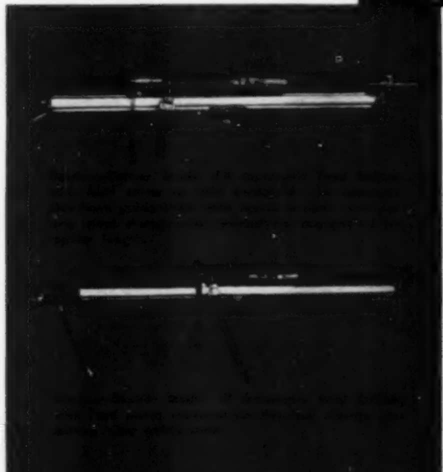
Increased demands for ore production from your mine?

A stepped-up development program to reach your ore reserves?

Hard rock you've got to "hole through" in a hurry?

A program to hold down production costs?

Gardner-Denver CF89H Automatic Feed Drifters.



For help in solving mining problems such as these, choose famous Gardner-Denver Automatic Feed Drifters. They're designed by men who know what it takes to do a fast, low-cost drilling job underground.

The Gardner-Denver self-adjusting feed, for example, responds automatically to bit penetration — is automatically regulated by the type of ground being drilled. Maximum drilling speed is easily maintained—"green" miners drill almost as fast as "old-timers." The long-wearing, "slow-motion" piston feed motor is economical to operate, too—uses only 3% to 5% of the total air consumption of the drill.

SINCE 1859

GARDNER-DENVER

Gardner-Denver Company, Quincy, Illinois
In Canada: Gardner-Denver Company (Canada), Ltd., Toronto, Ontario

THE QUALITY LEADER IN COMPRESSORS, PUMPS AND ROCK DRILLS

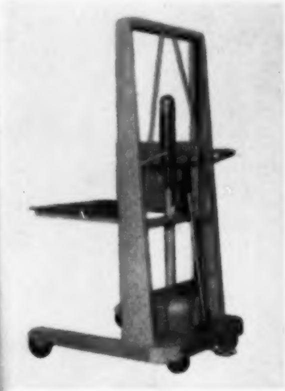
Manufacturers News

Steel Tape

The new *Lufkin* Royal Ni-Clad steel tape, made in 25, 50, 75, and 100-ft sizes, is stated to cost less than any other steel tape on the market. Figures and graduations are an integral part of the steel and the nickel plated surface is rust and corrosion resistant. **Circle No. 1**

Lift

A manually operated and propelled hydraulic lift marketed by *Century Products Co.* weighs only



150 lb, but the 23x24 in. platform will raise 1000 lb from floor level to 4 ft height. **Circle No. 2**

Scintillation Counter

The Scintillator from *Precision Radiation Instruments*, stated to have 100 times the sensitivity of the Geiger counter, is suitable for prospecting and may be operated from plane or moving vehicle. The instrument is contained in a 2-lb probe and the belt mounted battery box weighs less than 4 lb. **Circle No. 3**

Generators

Wincharger Corp. announces development of a Speedy-Shift wheelbarrow base type attachment for model 700, 1800 and 5000 engine-generator sets. Portability of the engine-generator sets themselves has been improved by belted construction for higher operating speed and maximum weight reduction. **Circle No. 4**

Cutter

A hand operated head cutter for carbide, cyanide, and other one-time drums of 24 to 30 gauge steel is produced by the *M. A. Schinker Mfg.* Co. The cutter leaves a safe, turned-in, flange. **Circle No. 5**

Scoop Shovel

The Scoop Shovel, new product from the *Thew Shovel Co.* for underground or surface loading operations, consists of a front end attachment applied to any basic full-revolving

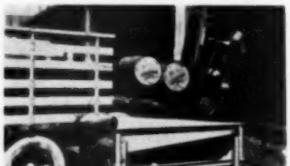
TL-25 turntable. The attachment is interchangeable with standard shovel, crane, clamshovel, dragline, and hoe attachments. It is equipped with 1¼ dipper in place of the ¾ yd dipper used on the standard TL-25 shovel. It is shown working in Oklahoma lead mine. **Circle No. 6**

Pump

A centrifugal pump introduced by *Gardner-Denver Co.* is primarily for circulating cooling water in air compressors, engines and other water cooled machines. Easy installation, simplified piping design and adaptability to various drives are claimed. Capacity is 67 gpm. **Circle No. 7**

Loading Ramp

A portable loading ramp to enable one man to perform the work of several by using push button control has been marketed by the *J. B. Illo Engineering Co.* The ramp may



be manually or electrically powered. Installed in five minutes at any loading dock, its hydraulically operated deck adjusts 24 in. to suit truck bed height. **Circle No. 8**

Spacers

Cut lengths of plastic tubing, made by *Irvington Varnish & Insulator Co.* serve as spacers in underwater dynamite charging. Users report the spacers reduce dynamite consumption by up to 30 pct while retaining effective breaking. **Circle No. 9**

Tachometer

To fill demand for wide-range tachometers *O. Zernickow Co.* has placed on the market two new units, with accuracy guaranteed to 0.5 pct at any reading. **Circle No. 10**

Torque Converters

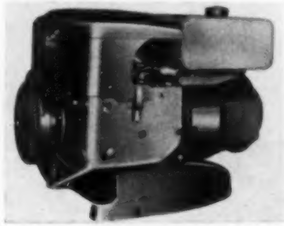
Caterpillar Tractor Co. announces factory-installed torque converters for 70 to 500 hp industrial engines. Used to power excavators, cranes and drill rigs, the converters allow high continuous output, and facilitate picking up loads. **Circle No. 11**

Dynamic Braking

The *Baldwin-Lima-Hamilton Corp.* recently made first application of dynamic braking to a 100-ton diesel-electric switcher. The 2-truck, 4-axle, 800 hp locomotive was delivered to the Medford Corp. for lumbering service in Oregon. **Circle No. 12**

Magnet Generators

An electric generating plant specifically for industrial lifting magnet use is built by *D. W. Onan & Sons*



Inc. in 3000 and 5000 watt dc sizes for stationary or portable service in manual or remote electrical starting models. **Circle No. 13**

Welding Aid

Weldaluminite fusing agent and deporosite extends application of inert arc welding to rimmed steel, producing porosity free welds. Made by *Spekaluminite Co.* the welding aid is sprayed or brushed on the joint to be welded and allowed to dry before welding. **Circle No. 14**

Truck Crane

The P&H Miti-Mite, *Harnischfeger Corp.* truck crane, was introduced after more than five years development and testing. Designed exclusively for truck service it is readily convertible for any front end service, shovel trench hoe, dragline, clamshell, or crane. It has a ⅜ yd cu shovel capacity and 7-ton crane capacity. P&H hydraulic control is operated from a full vision cab built for operator comfort. **Circle No. 15**

Free Literature

TUNNELS TO BINS. Originally designed for tunnel work, the strong lightweight liner plates used to construct this 30-ft diameter, 31-ft high, aggregate storage bin have several advantages. Bins and conveyor to tunnels can be quickly erected in any season, for permanent service, or to be salvaged and reused. Armco Steel Corp.

(16) ROLLER CHAIN: Engineering Data Book No. 2457 published by *Link-Belt Co.* is a comprehensive 148-page volume on roller chain and its application. Detailed engineering information covers installation, maintenance and lubrication of chains for drives, conveyors, and sprocket wheels. The book covers selection of drives to meet both standard and special installations.

(17) LABORATORIES: The *Fisher Scientific Co's.* line of 21 basic units of standardized steel laboratory furniture is described in catalog no. 25. The Unitized line was designed to combine the advantages of custom installation with the convenience of selection, delivery and continued flexibility that goes with ready-made units.

(18) PUMPS: Two similar, but functionally different types of vertical centrifugal wet-pit pumps; a heavy duty bilge pump for handling solids-free liquids and drainage; and a screenless sewage ejector, constructed to handle sewage and solids carrying liquid; are illustrated in a bulletin from *Yeomans Bros. Co.*

(19) TRACTORS: Maintenance practices are simplified for operators of Cat DW21, DW20 and DW10 tractors in a multi-colored cartoon-type booklet issued by *Caterpillar Tractor Co.* It will be published in French, Spanish and Portuguese, as well as English. Using scenes familiar to the operator, the booklet outlines factory-prescribed techniques in caring for wheel-type tractors.

(20) SAFETY APPLIANCES: *Mines Safety Appliances Co.* has an illustrated pocket-size booklet designed to serve as a token of welcome for visitors and give information on the company, its officers, plants, offices, and warehouses.

(21) INGERSOLL-RAND: "Our Latchstring is Always Out" presents a brief history of the company, its plants and products. Nearly 100 pictures show I-R's products at work; the story of the first practical steam rock-drill is told; and there is a list of milestones of progress.

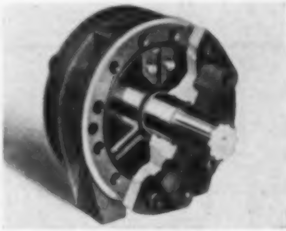
(22) SHEAVES: A four-page leaflet from *Allis-Chalmers Mfg. Co.* gives dimensions and engineering information on wide-range stationary control Vari-Pitch sheaves equipped with Magic-Grip bushings. This bushing provides quicker and easier installation and removal, and is 20 pct lighter than the former style.

(23) CONTROLLERS: Two bulletins from *Brown Instrument Div., Minneapolis-Honeywell Regulator Co.*, describe ElectroniK strip chart program controllers, used for time-temperature control programs, and cam operated program controllers in electric contact, electric proportional and pneumatic control forms.

(24) ALLOY CASTINGS: Of interest to users of stainless and heat-resistant alloys in cast form, a 48-page publication of *International Nickel Co.* illustrates industry application of typical alloys with over 175 photographs. Charts compare creep strength and corrosion and oxidation resistance. Tables of composition list principal alloys.

(25) STORAGE: "The Science of Economical Shelf Storage" by the *Frick-Gallager Mfg. Co.* discusses benefits of Rotabin rotary shelving. Savings of up to 50 pct in floor space are claimed for this system together with reductions in handling costs.

(26) ACTUATORS: Hydromotor rotary reciprocating actuators for air, oil, or other fluid medium are now the products of a new division of the



Bonnot Co. as a result of their recent purchase by Hydromotor Inc. A bulletin gives data on standard and special models.

(27) SCRAPERS: A folder on their 15.5 cu yd scraper released by the *Euclid Road Machinery Co.* describes major design features and provides detailed specification.

(28) EARTH MOVING: Literature from *Gar Wood Industries, Inc.* covers excavators and trailer hoists and bodies. A folder shows cam and roller and telescopic hoists as well as trailer bodies specially designed for these hoists. The largest hoist handles a 30-ton payload. Three specification booklets detail information on ¾ yd model 75A and 75B excavators and the 75BT truck crane. Capacities, ranges, dimensions and power-pants are listed.

Mining Engineering **October**
29 West 39th St.
New York 18, N. Y.

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1	2	3	4	5	6	7	8	9	10
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21	22	23	24	25	26	27	28	29	30

Students are requested to write direct to the manufacturer.

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Company__________

Street__________

City and Zone__________ State__________

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FOR BEST RECOVERIES

Frothers

HERCULES YARMOR® F PINE OIL
Standard of quality among pine oil frothers since the early days of flotation. Yarmor F Pine Oil produces a strong froth of good volume, satisfactory texture, and excellent cell-life stability.

Collectors

RADA (Hercules Rosin Amine D Acetate)
Siliceous mineral collector that has good solubility and handles easily.

Rosin Amine Derivatives: Special purpose collectors; uses vary with type of materials to be floated.

Technical information on these and other flotation aids is available. Send for 16-page booklet "Flotation and Hercules Flotation Agents."

HERCULES Flotation Agents

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The importance, validity, and acceptability of the Paley Report has become the great issue of the minerals industry. A 12-man panel made up of representatives of various facets of the industry sat down at the 1952 Mining Congress Show in Denver in an attempt to clarify the many opinions currently held concerning the report.

Howard Young, deputy administrator of the Defense Materials Procurement Administration, noted that the report had caused more discussion and serious thinking than any other government report. He did not subscribe to the "have not" principle, however, but called it a "need more" principle.

Horace M. Allbright, president of U. S. Potash Corp., believes industry should continue to develop the report. He cited the editorial appearing in the September, 1952 edition of Mining Engineering, "Don't Let It Die," as expressing his feelings.

Edward Snyder, president of Combined Metals Reduction Corp., says the statistics used in the report were used to promote a false international policy. He sees the report as a master plan for the development of the super-state. He also stated that the ore reserves given in the Paley report represent only 10 pct of what is actually available.

Arthur H. Bunker, president of Climax Molybdenum Co., and a member of the Paley Commission, believes that self sufficiency has never existed. Bunker feels that the national objective should be to obtain the most readily available material at the lowest possible cost.

Senator Henry W. Dworshak, of Idaho, offered the closing of the antimony mine at Stibnite, Idaho, as an example of using foreign sources in place of domestic supply. He does not feel the U. S. should stimulate foreign production.

Donald McLaughlin, president of Homestake Mining Co., terms the report timely and valuable, but that the trends outlined by the report are not necessarily conclusive. He sees a continuing growth of dependence on foreign sources, however.

Andrew Fletcher, president of St. Joseph Lead Co., forecast the deficits in lead production would not amount to more than 200,000 tons by 1975, as against the 900,000 ton prediction of the Paley Commission. He added, however, that no one can foresee our economy in 1975.

Simon D. Strauss, vice president of American Smelting & Refining Co., said "a Government-operated international buffer stock... would hang over the metals markets like a sword of Damocles. It would be intolerable for private capital to work in a climate in which the entire investment is in constant jeopardy because of potential political decisions of an organization responsible to no single nation."


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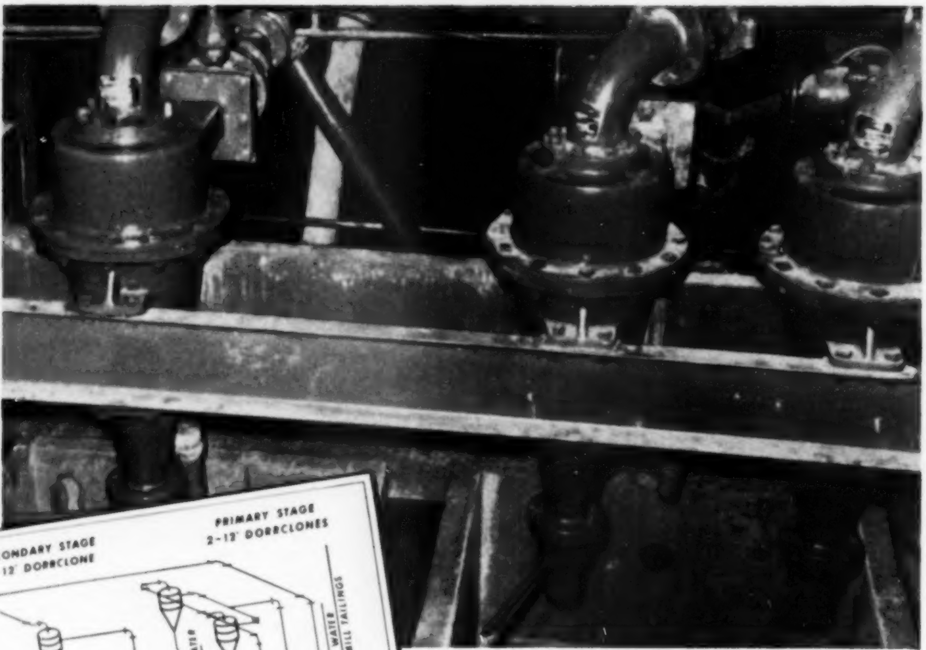

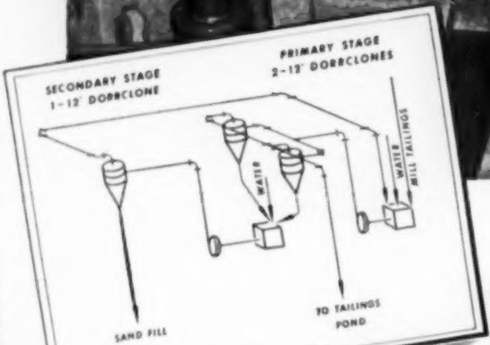

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Madsen Red Lake Gold Mines uses two stages of DorrClones for maximum flexibility*

From a 3" test DorrClone to the installation of three 12" dia. units. That's the story at Madsen Red Lake, one of Canada's most modern gold producers. Using a two-stage flowsheet as shown in the accompanying sketch, the DorrClones are producing 120 tons per day of fill with a percolation rate of 4-6 in/hr.

Key to this two-stage DorrClone setup is flexibility. While the units are in continuous operation, simple adjustments can be made to produce any type of fill from practically any composition of mill tailings. To illustrate this flexibility, three tests were made at Madsen Red Lake yielding the following results:

*T.M. Reg. U. S. Pat. Off.

	TEST A	TEST B	TEST C
New feed to primary stage, TPD	330	298	330
% solids to DorrClones	......	21.2	21.4
Sand Fill Produced, TPD	251	167	120
% solids	49.5	57.0	62.5
Percolation rate, in. per hr.	2.75	6.75	11.5

(Tests were made consecutively without shut-down)

Regardless of your particular mining operation, tailings composition or fill requirements, The DorrClone is an ideal tool with which to solve backfill problems. Write for a copy of bulletin No. 2500 to The Dorr Company, Stamford, Conn., or in Canada, to The Dorr Company, 80 Richmond Street West, Toronto 1.

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Mine, Mill to Launch Organizing Drive Soon

The 48th convention of the International Union of Mine, Mill and Smelter Workers, held in New York City recently, saw delegates vote funds for a new organizational drive expected to begin immediately.

John Clark, union president, told members in his keynote speech, that the union would be in a better position to make demands if it extended its membership to all phases of the nonferrous industry. He told the 250 delegates that union leaders have already laid extensive groundwork for the campaign. Clark said:

"I am proposing here that we concentrate on organizing during this coming year. I am proposing that we throw a heavy amount of resources in organizing. That means financial resources as well as resources of staff."

Strike Threat

Clark noted that the threat of strike brought increased wage grants from the major mining companies. He named Phelps Dodge Copper Co., Anaconda Copper Mining Co., and American Smelting and Refining Co. Settlements, according to Clark were based on 8¢ plus significant fringe benefits. He noted that the wage increase was the seventh gained by the union since World War II.

Referring to current negotiations with Kennecott Copper Corp., Clark said that the union would not be sidetracked. "This organization (the union) will make it come to terms." The union rejected a Kennecott bid.

Following a speech by William DuBois on trade unions in the colonies of various nations, Clark said that the aim of the IUMMSW was organization on an international basis. Chilean labor, he said, must be organized because at the present time they are actually in competition with American workers.

Grossed $900,000

On the financial side of the week-long convention, Maurice E. Travis, union secretary-treasurer, reported total union assets, as of June 30, amounted to $106,654 compared with $91,595 last year. Total income from local unions was more than $900,000. Almost $813,000 was spent on operating expenses, with salaries taking $90,000.

During most of the convention, Senator Pat McCarran, Democratic chairman of the Subcommittee on Internal Security, came in for much vituperation, both from union leaders and guest speakers. Four IUMMSW officers have been summoned before the Senate group. Senator McCarran was termed "an enemy of American labor."

The dredge "Caribbean" starts deepening the waters of the Caroni-Orinoco-Cano Macareo channel, over which the Orinoco Mining Co., U. S. Steel Corp. subsidiary, will ship iron ore from its Cerro Bolivar mine in Venezuela to the United States. Ore will be transported by rail to the port over an almost entirely downhill route. Shipments are expected to start in 1954.

Orinoco to Pay Duty On Imported Rail Ties

The United States Steel Co. subsidiary, Orinoco Mining Co. will have to pay $1 million in import duties on rail ties to be used in connection with its iron ore concessions at Cerro Bolivar, according to reports.

Orinoco needs the 400,000 ties to complete the 100 mile road from the mine site to the river port of Pureto Ordaz, under construction by the company on the Orinoco River

Company sources say that tests have been in progress since 1948 to find a suitable native wood, but they have proved unsuccessful. Some were found that could be used if they were creosoted, but a great deal of delay would be experienced before a treatment plant could be set up.

The company claims that local dealers could not supply creosoted ties in quantity or quickly enough to prevent construction delay. The mining company asked the government to set aside the import duties in accordance with the contract which states that no duty shall be charged for material the company is forced to import because of lack of available supply. Venezuelan dealers fought the move. The government ruled in favor of local timber dealers.

Climax Molybdenum Co., officials inspect the 60 in. Symons Primary Gyratory Crusher for the mining company's new Storke Level crushing plant. From bottom right up are: W. J. Coulter, vice president; M. Dessau mill superintendent; and C. J. Abrams, manager. A. Stazicker, Climax purchasing agent is between R. R. Shafter, top left, and H. M. Zoerb, bottom left, Nordberg Manufacturing Co.

Chemical Firms Extracting Uranium from Phosphate

The Atomic Energy Commission announced that four chemical firms will undertake to extract uranium from phosphate rock. The first company to start work is the Blockson Chemical Co., near Joliet, Ill.

The three others, now building recovery units, are International Minerals and Chemical Corp., and Texas City Chemicals, Inc. The AEC said the uranium occurs as a minor part of phosphate deposits in Florida and several Western states. The commission said that tests have revealed that when phosphate rock is processed into sodium phosphate chemicals and fertilizers, it is possible to extract uranium at the same time.

Recovery at Blockson is on a contract basis. The commission did not give production costs or how much uranium could be extracted.

U. S. Plans Asbestos Mill for Arizona Mines

In an attempt to encourage production, the U. S. government is considering establishment of a mill and purchasing facilities for chrysotile asbestos in Arizona.

The material has been mined on a limited scale in the U. S., but the nation is almost entirely dependent on foreign sources for high grade, long fibre, low-in-iron asbestos, known to exist in many places in Arizona. The contemplated program is aimed at speeding up production from known deposits and as an incentive to exploration.

According to Jess Larson, Defense Materials Procurement Administration head, the main reason for the lack of activity in known Arizona deposits at the present time is the lack of processing facilities. It is planned that DMPA will contract with a private company for the establishment of a modern processing plant for treatment of raw materials to meet stockpile and industrial needs.

Chrysotile asbestos occurs in Arizona in several localities over an area 60 miles long and 25 miles wide in the Salt River and Cherry Creek Basins, Gila County. Hard surfaced roads provide transportation to the area, and access roads connecting to the main highways reach most of the mines. Access to some of the undeveloped claims is difficult.

Arizona asbestos occurs in serpentine associated with dolomitic limestone intruded by diabase. The veins are confined to more or less horizontal zones several inches thick. They may lean out within short distances, or become richer.

A group of asbestos producers in the Globe area recently formed the Arizona Asbestos Producers Association. The association will study market conditions and problems, transportation improvement, and other factors in asbestos production.

Find Second Bauxite Deposit in Upata Area

Another large bauxite deposit was reported discovered in the Upata region of Boliva State by the Geology Department of the Venezuelan Ministry of Mines and Hydrocarbons.

The deposit is located at Cerro La Mesa and is reported to contain several million tons of the aluminum producing material. Geology Department engineers are continuing exploration to determine the exact size of the deposit.

A few months ago, the Government reported another bauxite deposit in the Upata region. Located on Cerro El Chorro, it was estimated to contain 2 million tons of bauxite.

Johns-Manville Plans African Asbestos Mine

Johns-Manville, operators of the world's largest Asbestos mine, in Canada, plans to develop another property in Africa. The company now owns two mines.

The new ore properties are located at Mashaba in the Victoria District of Southern Rhodesia, about 200 miles south of Salisbury and 120 miles east of Bulawayo.

Lakes Weather Key To Winter Steel Output

Great Lakes weather is the big "if" in the ore situation according to Defense Transport Administrator James K. Knudson. If Lakes weather holds good until December, most steel mills will be able to stay in operation through the winter.

Knudson praised the ore carriers for their record-breaking August effort, when 14,367,627 tons were moved. He summed up the present situation by saying:

"If freezing weather holds off on the Lakes until December, the nation can be grateful. For then, steel mill stockpiles, augmented by regular shipments through the winter over an all-rail route and by shipments of imported ore, may be sufficient to keep mills in continuous operation."

He said DTA "still hopes to see at least 76.6 million tons moved before the Lakes freeze over." The original goal of 96 million tons went by the board when the steel and iron ore strikes tied up the industry.

Bureau of Mines Drills Weyerhaeuser Property

A U. S. Bureau of Mines Crew is drilling on the old Weyerhaeuser copper property in Douglas County, Wisc., 60 miles southeast of Superior. No significant copper production has ever been reported in Wisconsin.

The Douglas County deposits are reported to be similar in mineralization to those in the northern Michigan peninsula. The Bureau crew plans to drill from two to four holes in the D lode area to substantiate the claims of previous industry drilling. In addition, they are seeking enough core to ascertain the grade and extent of mineralization.

Preliminary studies of the Douglas County property were made in 1943, and extensive trenching was carried on in 1944 and 1945. Copper was first reported in Douglas County in the late 1890's. Within the next 16 years some prospecting was done and a shaft sunk.

In following years a few thousand pounds of copper were mined, but no commercial production was recorded.

Truman Names 3 To Safety Board

President Truman named three men to the Federal coal mine safety board of review as provided for in the recently enacted Federal coal mine inspection law.

Named to serve on the board are: Alex U. Miller, retired U. S. Bureau of Mines official, who heads the board and represents the public; Charles R. Ferguson, acting safety director of the United Mine Workers of America, representing the mine workers; and Joseph G. Solari, assistant general manager of the Peabody Coal Co., Chicago, operator representative.

Rexspar to Explore British Columbia Claims

Rexspar Uranium and Metals Mining Co., of Toronto, Canada is planning extensive exploration of 90 claims three miles from Birch Island, B. C., and about 70 miles north of Kamloops.

Roads and camps are being built for a program which calls for 5000 ft of diamond drilling and surface stripping and mining of bulk samples for metallurgical testing. Limited exploration already carried out indicates uranium deposits with widths up to 90 ft and lengths of several hundred ft. Ore grade material has been found in one zone.

An appreciable quantity of rare earth elements are also reported present, rarely found in Canada. Franc R. Joubin, managing director of Technical Mine Consultants, Ltd., in charge of Rexspar development, estimates indicated tonnage possibilities in the fluorite zone in excess of 1 million tons. Open pit mining has been suggested for the property.

Two main zones of radioactivity were exposed by prospecting and surface trenching. Gieger drilling of all new and old holes will precede underground development. The initial phase is estimated to require eight months and about $75,000.

German Iron Ore Production Increases

Iron ore production in the Salzgitter area of the West German Federal Republic increased more than 25 pct from July 1, 1951 through June 30, 1952, over the same period the previous year.

About one out of every three tons of ore smelted in Western Germany comes from the Salzgitter-Peine area. Substantial production gains were reported by the Reichswerke AG fuer Erzbergbau und Eisenhuetten when the Watenstedt GmBF put its third blast furnace into operation recently.

Larson Reports Nicaro Now In Full Operation

Nicaro, the U. S. government-owned, privately-operated Cuban nickel plant, is now in full operation, and according to Jess Larson, Administrator of General Services, producing more nickel than during its World War II operations.

The plant is operated by Nickel Processing Co., a management corporation, owned jointly by National Lead Co., with a 60 pct interest, and Fomento Minerales Cubanos, a Cuban firm. Located about 450 miles from Havana, in Oriente Province, the plant produced 63½ million tons of nickel by 1947, when it was shut down as a surplus operation. The plant cost $32.5 million.

Restoration began in 1951 and cost $12 million. Larson said that full production was reached in mid-July. The operation has been adjudged 8 pct higher in efficiency than the operation of the average month in 1946. Larson added:

"With Nicaro in full operation, the free world is guaranteed a supply of nickel at least 10 pct larger than when the Communists invaded Korea."

The 12 Herreshoff furnaces are in operation for the first time in nearly five years. They were brought progressively into use in pairs. In July, the last pair was heated and placed in production.

Seeks to Reopen Reservation to Mining

Senator Harry P. Cain (R. Wash.) has recommended that 818,000 acres of unallocated land in the south half of the Colville Indian reservation be returned to the public domain and reopened to mineral entry.

Mineral development of the area has been prevented by an almost chaotic situation involving the Indian tribes and land taken by the Government. The Indians have filed claims for damages. Senator Cain said that the move for compensation makes it unnecessary for the land to be returned to the tribes as recommended at the last three sessions of Congress.

"I believe these Indians now have proceeded properly and legally in their attempt to recover compensation for the damages which they have sustained," Cain said. "If the claims are sustained by the commission, they will be justly compensated."

Reopening of the Colville reservation to mining has been one of the long standing objectives of the Tri-County Mining Association, with members in Ferry, Okanogan, and Stevens Counties, Wash.

Advance Tungsten Ore Goal Again

Defense Production Administration has revised upward the goal for Tungsten ores by setting a target of 40 million lb of contained tungsten per year by 1954.

The current figure is 6 million lb more in the annual rate than the one set August 1, 1952. It calls for an increase of 31 million lb per year over the 1950 rate of new supply of 9 million lb. The new goal does not apply beyond 1955, when a review will be made to ascertain the supply situation.

Domestic tungsten production in 1952 is estimated at 7.6 million lb, which means that the great portion of the ore will have to come from abroad. The Defense Materials Procurement Agency purchase program, announced April 21, 1951, called for a total of 1,468,750 short ton units. In July 1951 the figure was increased to 3 million short ton units by July 1, 1956.

Under the DMPA program, the producer must offer his output to industry at $65 per ton, if he is unable to sell at that price DMPA is committed to buy at $63 per ton.

U. S. Mining Firm To Reopen Malay Mine

The American Mining Co. will reopen the 1000 acre Temangan iron ore mine in Kelantan State, Malaya, and will ship ore to Japan at an annual rate of 500,000 tons.

The company will spend $5.6 million for barges and machinery in Japan and for dredging a ship loading point on the Kelantan River.

New Mining Rush In S. D. Black Hills

The Black Hills of South Dakota are once again the scene of a mining rush, but this time it's not for gold. Beryl, lithium, mica and other essential minor minerals found in pegmatite hold the attraction.

Bureau of Mines and other research laboratories are laboring to develop practical recovery processes. Men are in the field in search of new reserves of critical rare materials which have assumed importance in recent years.

Eleven pegmatite exploration contracts involving more than $195,000 have been signed between Government and mine operators in the search for the minerals. Domestic beryl mining has always been incidental to the recovery of other minerals within the pegmatite. Beryl is handsorted from the pegmatite along with feldspar, mica and lithium minerals.

The Black Hills are the nation's chief source of beryl. According to the U. S. Geological Survey the Black Hills contain 12,000 tons.

May Lease Ontario Iron Ore Properties

Stockholders of the Central Sudbury Lead-Zinc Mines Ltd., and Excelsior Mines Ltd., an associated company, were taking under consideration the leasing of 54 Sudbury district, Ont. mining claims as an iron mine prospect.

The lease would run 99 years and be granted to C. C. Huston, a mining engineer. The claims are located approximately 100 miles northeast of Sudbury. Huston is said to be acting in the interests of a Canadian steel company. Center of interest is the magnetite ore in two groups of the involved claims.

The two mining companies are to receive 15¢ a ton on all iron ore shipped or treated and compensation for removal of any other minerals. Significant sulphur content is believed present in the ore.

Marshall Plan Steel Aid Totals $220 Million

Since the start of the Marshall Plan in 1948, the U. S. has supplied more than $220 million for 31 iron and steel projects in nations taking part in the program.

Total cost of the projects was originally estimated at $864 million. Thus ECA-MSA supplied about one-fourth of the total cost. The remainder of the funds came from the individual nations. The projects were in the main finishing plants, rather than steelmaking facilities. The figure representing U. S. participation is for major projects only. It also does not include indirect purchases of equipment and materials.

Seek Germanium From Appalachian Coal Seams

The Pennslyvania Coal & Coke Co. has announced that it will undertake an exhaustive cooperative investigation of all mines and coal seams in the Appalachian region in search of recoverable germanium.

Germanium is now in greatly increased demand because of its use in the new transitors and in pure metal form is worth about $350 per lb. Concentrations of germanium, research has shown, is found in the organic matter of coals. Geologic surveys conducted in West Virginia revealed presence of appreciable quantities of germanium in several of the state's coal seams. Similar findings are hoped for in other Appalachian states. The top and bottom 6 in. of the seams have been found to contain the most germanium.

Pennsylvania Coal & Coke is sending instructions to producers on how to take sample blocks of coal for testing. No obligation is assumed by the producer and spectrographic analytical results showing germanium content of coal will be returned to the sender.

To date, prospecting has been limited and under the guidance of competent professors. The coal industry program is the first major breakthrough for wide and extensive exploration.

Whether a question of proper ventilation or the most economical method of transportation you will find the answer in Jeffrey engineered equipment for Metal Mine operation.

For instance, the Jeffrey 12-A AERODYNE Fan operates at a lower speed yet still maintains the high efficiencies characteristic of these ventilation units over many years. And for low cost transportation Jeffrey offers sturdy, long-lived Locomotives (trolley or storage battery), Belt Conveyors and Shuttle Cars depending upon the nature of your operation.

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Type 12-A Jeffrey AERODYNE Fan

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CARLOS IBANEZ DEL CAMPO is the new president of Chile. His election was felt in the copper industry almost immediately. The first reaction came on the New York Stock Exchange, where one company dropped eight points in a single day's trading.

But stock market reaction in this case may have been premature. Whether Campo, who is an avowed nationalist, will nationalize the mines is something that only a select few in the new government know. He has hinted that Chile will ultimately take over ownership of the copper mines, which provides about 33 pct of America's supply. The industry also accounts for about 70 pct of Chile's dollar income.

Campo was Chile's dictator from 1927 to 1931. During that time he was on good terms with American business enterprise. American operated electric and telephone companies were formed during his first regime. One unknown quantity enters into the situation, however.

The unknown factor is Campo's friendliness with Argentina's Peron, who can never be accused of carrying on a courtship with the United States. If Chile's new head goes down the line with his Argentine friend, U. S. copper interests may find themselves outside looking in. But Campo is said to be an independent man, not given to taking the lead or advice of others. Chile also needs American dollars. The $200 million a year she realizes from sales of copper to the U. S. is an important part of her fiscal well being—and not easily thrown aside. In addition, the American companies operating the Chilean mines have a negotiable, if somewhat intangible item on their side—experienced, trained personnel. Chile is short of the men who can run the mines, and is not likely to forget it.

Some Campo supporters campaigned on a platform of nationalization and the voters may have taken it to heart. The tipoff may come in the form of a strike—one that will force Campo to do something he may not entirely want to do.

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IN the meantime, elsewhere in South America, nationalism marches on apace. Bolivia is set to nationalize its tin mines—and soon. President Victor Paz Estensoro has promised that nationalization will be a fact by early October. The committee charged with the study of nationalization makes its report soon, and Estensoro has said:

"I can state that nationalization will be a fact by the end of that period, whatever obstacles may be raised, either inside or outside the country."

Bolivian mines are operated by European and American engineer personnel, although they are owned principally by three Bolivian firms. Bolivia faces much the same problem that Chile does. Its easy to pass a law, but its tough to find people who can make the law mean something.

In the U. S., little can be expected in the way of progress in the contract talks between the U. S. and Bolivia. The contract is no nearer formulation than at any time in the last year.

ALCOA seems to have stolen a march in the race of cheaper power for the aluminum industry. Its proposed Alaskan project appears to be rooted in the cheap hydroelectric energy which will come from Canadian water power sources. Power will be generated by damming the Yukon River at Miles Canyon near Whitehorse. Several legislative barriers remain in the way of the Taiya Valley undertaking. First, Canadian Government permission will be needed for obtaining water power. Second, the project requires about 20,000 acres to accommodate the Alaskan smelting facilities and the electric power developments needed to operate them. Presently, there is no means under the law for Alcoa to acquire that much land. Thus, Congress will have to pass special legislation if the project is to go through to completion.

Alcoa will finance the entire project privately, without guarantees of any kind from Government and without special accelerated amortization arrangements. The time needed for completion is estimated at approximately four years and will provide year round employment for about 4000 people when in full operation.

Alcoa has made some attempt to use power sources other than hydroelectricity. Along with several aluminum producing companies Alcoa has been using natural gas, but the cost factor has been far from easy to assimilate. Thus, despite the inconvenience of location, and the accompanying higher transportation costs, Alcoa may have decided that Alaska is the most logical area for future operations. One Alcoa executive summed up the situation with:

"The most important single ingredient in aluminum production is power. Hydroelectric power is the best and cheapest."

In the meantime Alcoa will continue experimenting with other power sources. Its Rockdale, Texas works, now under construction, will be the first aluminum plant to use lignite.

Alcoa officials believe construction of the smelting plant will start either during late spring or early next summer. Construction start will depend on obtaining U. S. Government approvals of land purchases and water rights from Canada. The Canadian Government has indicated that it is quite happy about the project.

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BLAST Furnace Must Go—Blast Furnace Obsolete, But Due to Remain—Indicate Sponge Iron Process May Gradually Replace the Blast Furnace. These headlines appeared recently in newspapers, coming from reports presented in Geneva for the United Nations Economic Commission for Europe. To the average observer, it would seem that a major technological change was at hand. To the technician, it was confusing.

A careful analysis however, tempers these items. The two major replacements reported for blast furnace pig iron were the low shaft furnace employing oxygen, and the sponge iron process. It cannot be disputed that the reports leading to the headlines are true, and the conferees were all outstanding men

in the industry. However, the reports apply to conditions peculiar to Europe, or more specifically, the country in which the report originated. The ores and fuels available in each country, as well as production rates, were the deciding factors. It cannot be inferred that a projection can be made for the conditions present in the United States.

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THE late steel strike resulted in an unusual reversal of field when Brazilian interests shipped steel to the United States to assure completion of a blast furnace under construction in Cleveland for the national Volta Redonda mill. The furnace is one of the key factors in an expansion program scheduled to raise Brazilian steel production by 280,000 tons, bringing annual output to 1 million tons. The first consignment shipped to Cleveland included 500 tons of slab and plate steel, and was followed later by another 300 tons. Brazil wants to blow in the furnace by July 1, 1953, and the restricted allocation system brought about by the steel shortage offered a real threat to the deadline. Volta Redonda, established during World War II by Brazilian and American financing, accounts for more than half of Brazil's steel production.

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SOVIET Russia recently took the rest of the world into her confidence and announced her industrial plans for the next five years. While no exact numerical goals are stated, the percentage figures indicate that this will be a maximum effort to close the gap between the USSR and the western capitalists. Even if she reaches the established marks, Russia will be a long way from equaling the free world. Indicated in the plan is a steel production increase of about 62 pct. Estimated soviet steel production for 1950 was 27.6 million metric tons, which would mean 44.7 million metric tons by 1955, when the latest five year plan matures. Russia also aims at a coal production of 374 million metric tons and a pig iron goal of 34 million metric tons.

Possibly the greatest single effort will be in aluminum production, with the plan calling for an increase of 176 pct. French economists estimate that 1950 production reached 200,000 tons. Copper is scheduled to rise 90 pct, zinc 150 pct, lead 170 pct, nickel 53 pct, and tin 80 pct. The labor force increase is expected to be only about 15 pct, which means that the USSR will have to intensify its use of every available manhour. Increase in automobile production will be only about 20 pct and tractors 19 pct, indicating that Russia will stress arms production over that of consumer goods. Yet, one of the stated aims of the new plan is reduction of prices to instrument raising of the buying power of industrial and farm workers. If consumer goods production is raised only a little, the increased buying power is meaningless. The betterment of standard of living can only be nominal—and not actual.

JOHN R. STEELMAN, acting Defense Mobilizer, told The President that recovery from the two-month steel strike is faster than Government officials had anticipated. "On the whole the production agencies feel that most industries are returning to their pre-strike levels of production faster than had previously been expected."

"The steel companies will be able to overcome a large part of their pre-strike backlog of military atomic energy and machine tool orders . . . "

Much of Steelman's optimism hangs on the belief that ore shipments will reach steel plants in sufficient volume by the time the Great Lakes freeze over.

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THE International Materials Conference has been talking out of two sides of its collective mouth when dealing with the problem of an adequate sulphur supply. The IMC says that since July 1951 sulphur consumption, 8.3 pct ahead of production in the first half of that year, has been brought into line, and the strain eased considerably. In almost the same breadth, the conference notes that U. S. requirements for the second half of 1952 will be 3.83 million tons, against an expected production of 3.2 million tons. Which set of facts is the more pertinent is a matter of conjecture. But the IMC is not alone in causing confusion. Several other factors have contributed mightily to muddle the situation.

Gulf Sulphur Corp. recently announced an important new sulphur discovery in Mexico. The strike will add somewhat to the world sulphur supply and to the growing sulphur industry of Mexico. But the Mexican Government has indicated that the new source may add to world supply—but not world availability.

Alfonso Cortina, minister and economic counsellor of the Mexican Embassy in the U. S. has recommended that Mexico concentrate on satisfying her domestic supply. In addition, he urges that Mexico become an exporter of processed or semi-processed sulphur, rather than the crude product. He would have a tight lid placed on exports, and although he seems to be making the recommendations on his own, they are believed to be the general outline of thoughts held by Mexican Government officials in a position to instrument them.

Current sulphur development in Mexico has not reached a state of actual significance, but undoubtedly in the future, she will be one of the primary sources of the product. The Paley report places its chief reliance on future sources south of the border.

What can be concluded from the conglomeration of conflicting reports is that the world sulphur shortage is easing—but not as fast as some would think. The demand trend is up. How much will be needed by say, 1955 is debatable. The free world grand total new capacity by that time will be in the neighborhood of 4,040,700 tons. But much of the new supply will be obtained through other means than the Frasch process—at what now may be considered a prohibitive cost.

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MINING ENGINEERING

EDITORIAL

RECOGNIZING THE CROSSROADS

BEING at the crossroads, metaphorically speaking, seldom has the advantages of the literal sense of the words. One seldom has precise knowledge of the existence of the metaphorical crossroads or forks in the road of life. A case in point is the Committee on Education of the Engineers' Council for Professional Development, which for twenty years has been accrediting the engineering curricula in the educational institutions of the United States. The committee is now at odds with itself and with the National Council of State Boards of Engineering Examiners over the accrediting of courses which in the opinion of many engineers are not strictly speaking engineering curricula at all. Such courses include: fire protection engineering, crystal engineering, nuclear engineering, and geological engineering.

The crossroads might be placed arbitrarily somewhere between 1936 and 1938 when the committee first started accrediting curricula and included the first course in industrial engineering. Purists might argue that the committee first wandered from the straight and narrow when it recognized subdivision of engineering into the major branches of mining, metallurgical, civil, mechanical, electrical, and chemical engineering.

Accrediting is certainly an important function because it provides a standard for engineering education and it should be done on a national basis by an unbiased group like the Education Committee of ECPD. Students, educational institutions, and industry derive great benefit from proper standards. The autonomous State Boards of Engineering Examiners have long accepted ECPD accreditation as evidence of the required academic training of engineers aspiring to licensing. But the State Boards today question the wisdom of applying the name *engineer* to graduates of some of the so-called fringe curricula. The Education Committee, having gone so far in accrediting some of them, has numerous applications pending for accreditation of similar curricula and even new courses never before accredited.

Arguments against recognizing highly specialized engineering curricula are numerous. The Salary Stabilization Board has disallowed industrial engineering from classification as engineering for its purposes. A graduate of a specialized course in geological engineering at a school in the Southwest slanted heavily for practice in the petroleum industry might not be interchangeable with a graduate of a similarly designated curricula at one of the institutions in the North Central states where the emphasis might be on metallic mineral deposits. Industry in general should not expect the universities to supply a degree in each of the specialized engineering classifications it might set up for purposes of salary or job designation.

Having gone so far, it is easy to understand the difficulty of the Education Committee. ECPD might well call a moratorium for the purpose of taking stock of its position. If the present trend is allowed to continue, it is apparent that an engineering institution could be subdivided into a multiplicity of departments teaching specialized curricula resulting in a subordination of the basic concepts of engineering.

Although the parallel of the medical profession is trite, it should be pointed out that degrees in medicine are not labelled by the numerous specialities practiced by doctors.

For practical purposes the ECPD would do well to accept the fact that the crossroads have been overreached and protect the engineering profession by coming up with minimum standards for an engineering curriculum.

Freezing Method Solves Problem In Carlsbad, N. Mex. Shaft

Freezing applied to shaft sinking can be the most economical and satisfactory method for penetrating water-suspended materials, such as the quicksands found in the Carlsbad area. Attempts to grout can fail if water content dilutes the cement or flows carry it away. PCA made two unsuccessful attempts to grout, and then was forced to the Poetsch freezing method, sinking to 450 ft without mishap.

by John E. Latz,

for the Joint Venture Group of Consolidated Western Steel Co., Missouri Valley Constructors, and Winston Bros. Co.

QUICKSAND far below the surface, stopped two attempts by the Potash Co. of America to sink a third shaft to a potash bed that lies 1000 ft below the southern New Mexico desert. Virtually all methods of grouting were employed in the two failures without any indication of consolidation of the flowing sand. Sinking the South Shaft at the Carlsbad potash mine seemed an almost insurmountable task, until freezing was employed as a final resort.

This process that proved to be the answer may be defined as "the sinking of a shaft by freezing, consists of the formation, in the water-bearing strata, of a large block of frozen ground in which it is possible to sink a shaft without danger of an influx of water, providing the ice-wall between the water outside and the shaft itself is strong enough to resist the hydrostatic pressure."

Freezing Method Used Extensively in Europe

Successful frozen ground shaft sinking projects had been completed in Europe, and Russel G. Haworth, resident manager of the Potash Co. of America, went abroad in search of the know-how needed for the New Mexico undertaking.

The freezing process, with a history of two failures at the time, was first successfully employed by Poetsch in 1885 at the Houssu Collieries near Haine-St. Pierre, Belgium. Poetsch contracted to sink the remaining 64 ft of an uncompleted shaft through water laden strata from a depth of 190 ft to a depth of 254 ft. This work was in progress until 1887. The freezing and sinking consumed more than two years.

As a result of early failures, the process was not favorably received until soon after the turn of the century. The accurate prediction of success in a given application was made possible by the concerted effort and study of a group of European engineers. Means of accurate deviation measurement and control were devised, the nature and properties of frozen water laden soil (the ice wall) were investigated, a thorough and rational basis of refrigeration load calculation developed, and concurrently the efficiency and dependability of refrigeration machinery was greatly advanced. Numerous shafts in Germany, Belgium, France, and England today testify to the reliability and economical surety of the freezing method.

JOHN E. LATZ is Mechanical Engineer for Consolidated Western Steel Div. of U. S. Steel Co., Orange, Texas.

The shaft collar as its forms were being stripped. Note the pipe from the center relief hole, and the frosted freeze hole connections.

Applied to the South Shaft, which is circular in cross-section and 15-ft diam, the freezing method consisted of drilling holes 350-ft deep, through the water bearing strata into impervious ground—the freeze hole pattern being concentric with the shaft and 31-ft diam. The holes were cased with pipe closed on the bottom end. Smaller tubes were placed inside these casings and connected to a header supplying chilled brine from a refrigeration plant. The brine descended in the tubing to the bottom of the hole and ascended through the annular space between the two pipes to the top of the casing; flowed into the return header and back to the refrigeration plant. Under these conditions, each of the freeze hole casings became surrounded by a cylinder of ice

that increased in diameter with time. The merging of the ice cylinders eventually formed a hard frozen wall within which the shaft was sunk.

Contract Awarded and Shaft Site Selected

Following the award to Winston Bros. Co. of a contract for engineering and construction of the South Shaft, exploratory drilling was begun on the proposed shaft site in June 1950. This drilling revealed the formations as logged in Fig. 1. The presence of quicksand was detected at several horizons between the surface and the top of the salt formation 440 ft below the surface. The most serious quicksand conditions appeared, however, at depths of 120 to 150 ft in solution channels in gypsum and again between 310 ft and 350 ft in sand and silt. There were no indications of the presence of water below 350 ft. In addition there were fissures and cracks in the two limestone members from which heavy flows of water would result unless these zones were frozen or grouted. The following was then apparent:

1—The freeze holes must extend from the surface to a depth of 350 ft.

2—The concrete lining of the shaft must go down to a depth of 450 ft, extending into the salt formation in order to effect the necessary water seal.

3—The lower portion of the shaft lining must be designed to withstand an external hydrostatic pressure of approximately 300 ft, since it was apparent from area subsurface geology that the various water zones were vertically connected.

Testing to Design Freezing Plant

With the information obtained by initial drilling, the freezing calculations were completed. These calculations yield:

1—An economical solution of the ice wall thickness required to withstand external soil and hydrostatic pressure.

2—Spacing of the freeze holes.

3—Required refrigeration capacity and temperatures. The principal variables of required ice wall thickness are the nature and properties of the ice wall itself.

The first tests on samples of frozen soil were conducted by Alby and published in "Annales des Ponts et Chausees," 1887, Vol. VII, p. 338. These tests reflected an increase of resistance to compressive and tensile destruction with a diminution of temperature. The resistance was also found to be proportional to the percentage of water contained in the sample. These tests, however, all of an industrial character, failed to reveal the plastic nature of the frozen soils as determined by Sauvestre and published in "Traverse dans le Creusement de Deux Puits D'une Assise de Sables Boulants," Liege, Charles Desoer, 1918. The degree of plastic flow varies with the time during which the external pressure acts, the diameter of the shaft, the height of the wall exposed, the thickness of the wall, its temperature and its constitution. The conditions of South Shaft, however, as mentioned later, necessitated a mean ice wall temperature appreciably lower than that required to produce a minimum ice wall thickness.

The circular spacing of freeze holes was 3 ft, the minimum allowed by drilling accuracy.

The determination of refrigeration load imposed by a given cooling rate, that is, the time allowed for initial formation of the ice wall, is complex. Saclier and Waymel in "Bulletin de la Societe de L'Industrie Minerale," Vol. IX, 1895, p. 27, rigorously formulated a rational approach, modifications of which yield reasonably accurate results. The assumption upon which this derivation is predicted is the existence of the condition of a frozen annulus of strata around each freeze hole, the annuli being bounded on the side remote from the shaft by what is in effect an infinite solid of more or less uniform temperature increment in depth, dependent on the geothermic gradient of the locale. The required refrigeration capacity thus formulated is a Fourier development which varies with: 1—The selected perimeter of the freeze hole circle; 2—the average coefficient of thermal conductivity; 3—the brine temperature; 4—the concentration and specific heat of the brine solution; 5—the water concentration and specific heat of the soil; and, 6—the depth of the freeze holes. The load imposed by the maintenance of the ice wall in a condition of unvarying thickness during the sinking period is, of course, considerably less than that imposed by the initial cooling rate. However, this latter load is by no means unchanging in its magnitude since there are factors varying with the depth to which the shaft sinking has progressed which in turn affect the refrigeration load.

Close-up of the connections between brine headers and freeze-holes. Thermocouples were inserted through the tees and through sides of the 6-in. casing.

The capacity demands thus determined for South Shaft were met by the selection of two 10x10 vertical, single stage, motor driven, ammonia compressors, each having a capacity of 56 tons, or a total of 112 tons, while cooling 1000 gpm of calcium chloride brine from 8½° to 5°F in one 35 in. diam x 12 ft long, four pass, nonpriming type evaporator. The selection of single stage compressors in the face of relatively high compression ratios was dictated by the more favorable delivery and greater salvage value of the single stage compressors as compared to two stage equipment. In addition, the greater simplicity of operation required favored the use of single stage compressors for the inexperienced personnel available. An evaporative condenser with heat dissipation capacity of approximately 1.75 million Btu per hr was selected in lieu of a shell and tube type condenser, because of the short supply of water and favorable wet bulb temperatures in the Carlsbad area. The temporary nature of the refrigeration installation warranted the use of a minimum of auxiliaries; thus the selection of appropriately sized oil separator, ammonia receiver, and suction trap completed the refrigeration equipment specifications.

The relatively steady refrigeration load permitted the use of inexpensive liquid level float controls for the admission of ammonia to the evaporator. Refrig-

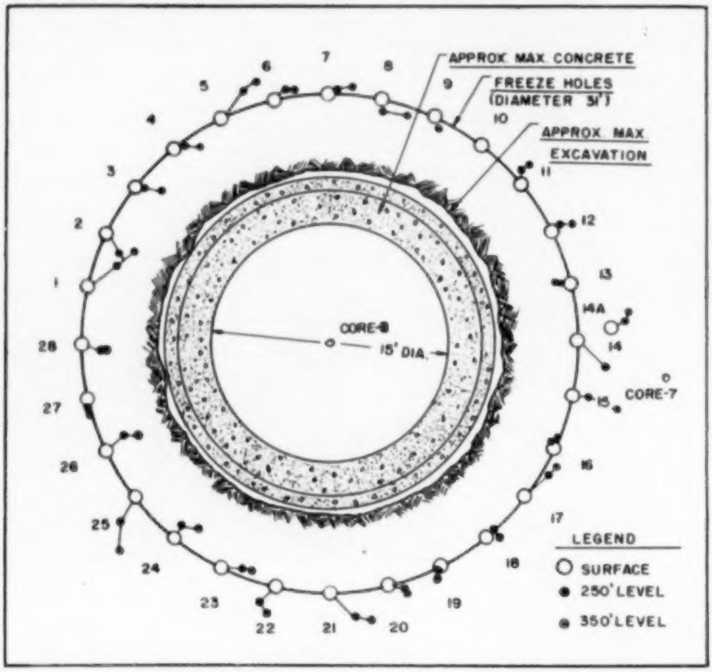


Fig. 2—This deviation chart shows clearly the difference in drilling accuracy between cable-tool drilled holes (1, 5, 13, 15, 25, 27 and 28) and the remainder, drilled by rotary rigs. Sperry-Sun directional survey determined need for hole 14A to supplement original pattern, since hole 14 intersected hole 7 at 260-ft depth.

eration capacity control was obtained by manually controlled suction pressure.

Drilling Freeze Holes

Drilling of the freeze holes began approximately July 15, 1950. Through the greater part of the drilling period, three drilling rigs were employed. Cable tool rigs were used initially, as they were readily available. However, due to their relatively slow progress and unacceptable accuracy, these rigs were soon replaced by rotary rigs. Holes numbered 1, 5, 13, 15, 25, 27, and 28 were drilled by the cable tool rigs, and the remainder of the holes were drilled by the rotary rigs. The rotary rigs drilled an 8-in. diam hole from the surface to the final depth. Deviation control during drilling was aided by means of a Totco recorder. This method of deviation measurement yields a nondirectional deviation magnitude which informs the driller of required adjustments in the variables of drilling weight and rotation speed.

Upon completion of each hole, it was immediately cased with 6-in. schedule 40 seamless pipe. This pipe was welded into 40-ft lengths before being lowered into the hole. The first length placed in the hole was closed on the bottom end by a welding cap. The succeeding 40-ft lengths were welded to each other while being held vertically by the rig over the preceding length. After completing the casing operation, the pipe was hydrostatically tested to 100 psi. The casing in hole No. 10 was the only failure. To remedy this condition, a 4-in. casing was placed inside the 6-in. pipe, and tested successfully.

To determine the directional accuracy of the drilled holes, a Sperry-Sun directional survey was run. As the result of this survey, it was deemed necessary to supplement the original pattern of freeze holes with one hole, No. 14-A. Hole No. 14 had been drilled to a depth of only 260 ft, at which point the drill bit encountered the casing in nearby core hole No. 7. Freeze hole No. 14-A was drilled to fill this gap at the 260 to 350 ft depth.

Following the drilling, casing and testing of the freeze holes, 2-in. schedule 40 tubing was set inside the 6-in. casing, being welded and placed in the same manner as the casing. The tubing, open on the bottom end, was lowered to within 18 in. of the bottom of the casing to provide an unrestricted flow of circulating brine.

Drilling the freeze holes and construction of the refrigeration plant were completed early in November 1950. The 8-in. brine supply and return headers were fabricated and installed. The supply header was connected to the top end of the 2-in. tubing and the return header connected to a side outlet in the 6-in. casing. Note that the headers and connections were depressed below normal grade and timbered over to permit movement of equipment and personnel during the shaft sinking period.

The regulation of chilled brine flow and consequent control of the freezing effect of each hole was done with manual valves in the lateral connections. Iron-constantan thermocouples were placed in the bottom and top outlet of each hole. The ice wall formation method employed was uniform simultaneous growth of ice around each freeze hole, requiring similar refrigeration effect in each hole. By means of the thermocouples and manual control valves, a uniform temperature difference between the bottom and the top of the hole was maintained.

Starting Refrigeration Plant

The refrigeration plant was started on Nov. 19, 1950, with brine circulation through five of the freeze holes, allowing a rapid lowering of the brine tem-

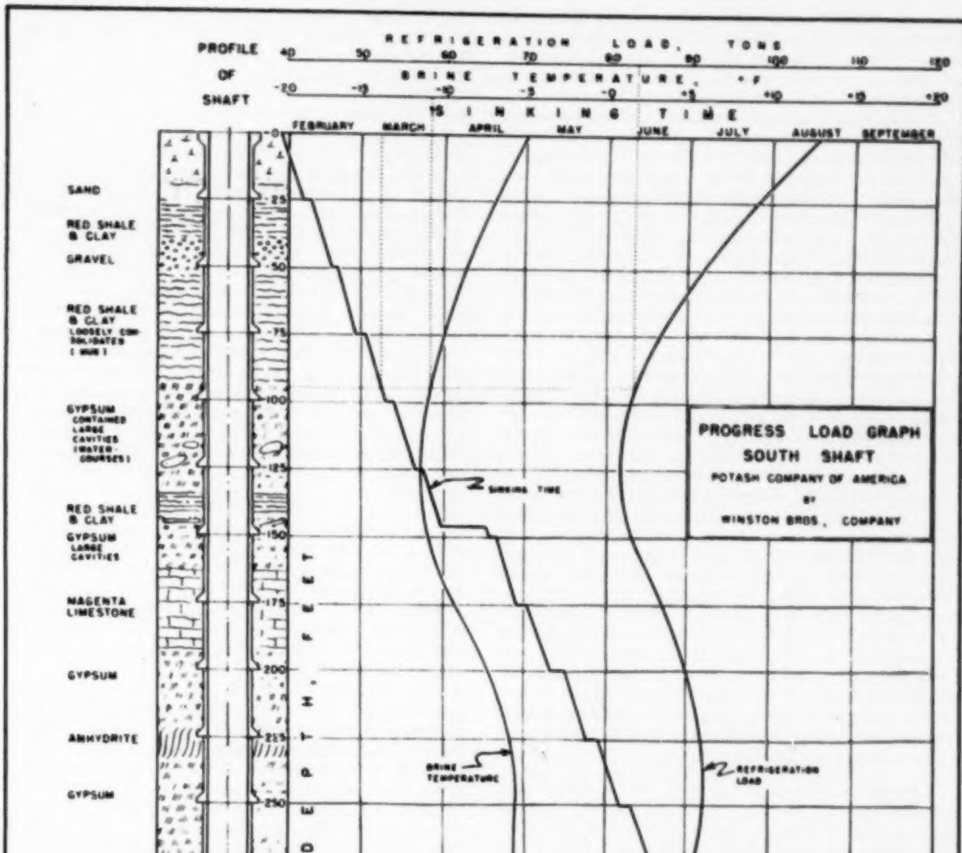


Fig. 1—Drilling logged the formations shown at left. The most serious quicksand conditions appeared at depths of 120 to 150 ft and also between 310 and 350 ft. On the sinking time line horizontal segments reflect time to prepare forms, to concrete, and to prepare for next 25-ft section.

perature and obtaining a steady refrigeration load for simplicity of operation. When a brine temperature of approximately 30°F was obtained, holes were opened one by one into the brine system until all 29 holes were operating by Nov. 26, 1950.

During the weeks required for the formation and closure of the ice wall, construction and installation of surface facilities necessary to the sinking operation, such as the concrete batching plant, hoist and hoist house, and headframe, were continued.

The steel headframe, designed and fabricated by Consolidated Western Steel, div. of U. S. Steel Co., at its Orange, Texas plant, was 90-ft high and of shop riveted and field bolted construction. The design of the headframe was based upon the 130,000 lb breaking strength of 1¼ in. diam plow steel cable. This permanent steel headframe will serve the mine after completion of the shaft sinking.

Closure of the ice wall was checked by observing the water level in a center relief hole. This hole was cased with perforated pipe and extended from the surface through the lower water zone. The water trapped within the cylindrical ice wall, upon closure, expands as a result of the decreasing water density consequent to the decreasing water temperatures, and so rises in the hole. The rising water was first noticed on Jan. 15, 1951 indicating that the ice wall had closed. From this time until sinking of the shaft was started on Jan. 29, 1951, the water that seeped into the center relief hole was pumped out in an attempt to remove all water contained within the cylindrical ice wall. This was necessary to minimize upheaval of the frozen soil at the surface.

Prevention of soil disturbances was deemed advisable to forestall stressing of the concrete lining when thawing the ice wall. As a result of this precaution, there was no noticeable soil disturbance.

Sinking the Shaft

Excavation of the first 25 ft of the shaft was performed with a crane and clamshell, and without the aid of explosives. Explosives were used, however, throughout the remainder of the shaft. Blast holes were 1⅜ to 2-in. diam and were drilled a maximum of 36 in. deep, dependent upon the formation encountered. A large number of such holes were drilled in a circular pattern to provide close trim and fine excavation material. Ten delay exploders were used to avoid undesirable shock on the lining and ice wall.

Excavation material was removed from the shaft in two manually filled buckets of 31 cu ft capacity. These were lowered and raised alternately by the permanent mine hoist. Neat excavation in the shaft was accomplished with pneumatic tools. This excavation was carried to a point several inches beyond the designed exterior of the concrete lining and 1-in. diam steel rods, 12 in. long were inserted in the earth wall to support the corrugated iron backforms, which were shaped to the desired circular conformation by means of angle iron rings, rolled and erected to a diameter of approximately 18 ft thus wedging the backforms between the angle iron rings and the steel rods. The annular space so formed between the soil and the backforms, was provided to prevent contact between the concrete lining and the subfreezing soil. This opening was later filled with grout, pumped through 1½ in. pipes formed in the shaft lining at regular intervals, again forestalling any soil disturbance that might occur when thawing.

Concreting the Shaft

In the sequence of sinking, the backforms and incidental pins and angles were installed as the excavation progressed. The shaft was excavated to the bottom of the 25-ft section of lining to be poured. The bottom of the shaft was drilled for the next round charged and exploded, but the loosened material was temporarily left in place. The bottom form ring was then set in place on the broken rock and levelled, after which the vertical reinforcing bars and the 10-in. rubber water stop section were placed. The upper ends of the vertical reinforcing bars were tied to the vertical bars projecting from the preceding section of the lining, and the lower ends of these bars projected through the bottom form ring to tie into the next section of shaft lining. The bottom ring also retained the rubber water stop. The interior circular forms were then installed from bottom to top, and the horizontal reinforcing bars placed concurrently.

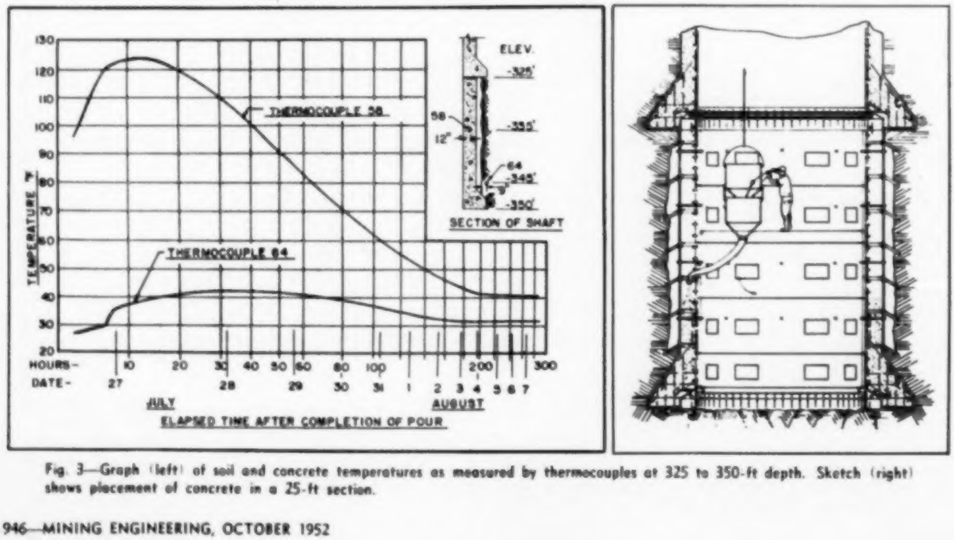


Fig. 3—Graph (left) of soil and concrete temperatures as measured by thermocouples at 325 to 350-ft depth. Sketch (right) shows placement of concrete in a 25-ft section.

Concrete, batched and mixed on the surface, was lowered in a ½ cu yd concrete bucket. A flexible hose coupled to the bucket, conveyed the mixture into the forms through hinged doors regularly spaced circumferentially and vertically in the forms. The concrete poured at the section joints was formed in the interior by means of a special closure ring.

The interior concrete forms were removed 25 hr after the completion of the concrete pour. During this period, the job personnel installed the safety ladders and crosshead guides in the preceding section of the shaft. When all forms, except the bottom ring had been removed the previously blasted material was excavated below the bottom ring, allowing the ring to be dropped below the ends of the projecting reinforcing bars.

Progress Load Graph

The progress of sinking is recorded in Fig. 1. The slope of the diagonal line of the curve labelled sinking time is the rate at which excavation proceeded through a given 25-ft section of the shaft. The horizontal segment at the section joints reflects the time required to place and set the reinforcing bars, concrete forms, and concrete in the section immediately above that joint, as well as the time required to remove the forms after the concrete had set and to prepare for the section below.

Also indicated on the Progress-Load Graph is the Brine temperature plot. This is the temperature at which the brine was supplied to the freeze holes. To illustrate the relations reflected by this graph, note that when the shaft had been excavated to a depth of 95 ft (at which time the concrete lining had been completed to a depth of 75 ft), on March 7, 1951, the brine temperature was −11°F and the refrigeration load was 83 tons. The refrigeration load and the brine temperatures continued to decrease from this depth to that of 125 ft. From a depth of 125 ft to that of 325 ft there was a relatively uniform increase in the refrigeration load with the depth of completed shaft. This increase amounted to approximately 1 ton of refrigeration with an increase of 25 ft in the depth of the shaft for a sinking rate of approximately 2-ft per day. The increase in load resulted from the dissipation of the heat of hydration of the concrete lining in the sections above a given completed depth, as well as the increase in heat transfer surface presented to the ventilation air. It is apparent, then, that for an increased sinking rate, the rate of load increase with respect to depth will also increase.

At a depth of 325 ft, one of the two compressors was shut down. As a result the refrigeration load imposed for the maintenance of a relatively constant brine temperature exceeded the capacity of a single compressor and the brine temperature increased uniformly with depth until the completion of the shaft lining on Sept. 13, 1951.

Brine temperatures lower than calculated were produced to insure the solidification of one zone of water, which in the latter stages of drilling was found to be relatively high in salinity. There existed the possibility that some of this saline solution might be entrapped within the ice wall. If this occurred, ice would precipitate from the solution, thus increasing the salt concentration and lowering the freezing point of the remaining solution. Consequently to prevent structural defects in the ice wall, these possibly entrapped solutions were solidified by lowering to this eutectic temperature.

Departed From Precedent Lining Methods

The lining of South Shaft with structural concrete constitutes a departure from lining methods previously employed in conjunction with the freezing process. To determine the concrete specifications necessary to insure a structurally sound lining, V. H. Montgomery, project manager for Winston Bros. Co., conducted laboratory tests in which actual conditions were simulated. In addition to these tests, thermocouples were placed in sections of the shaft, one in the concrete lining and one in the adjacent soil. The temperatures as measured in the section between the 325 and 350 ft depths. Dissipation of the heat of hydration was such as to maintain a concrete temperature in excess of 40°F for a period greater than 12 days after placement. This indicated there was no freezing of moisture and consequent spalling of the concrete.

Soil temperatures adjacent to the concrete during this period were elevated to a maximum of approximately 43°F, indicating that there was no appreciable thawing of the ice wall resulting from the heat of hydration of the concrete.

The freezing plant was shut down on Sept. 29, 1951, and as soon as the ground adjacent to the concrete shaft lining had started to thaw, grouting of the area between the ground and the concrete was started. The ground was allowed to start thawing prior to grouting to prevent freezing of the grout. All grouting operations were completed before complete thawing of the ground, to hold the grout and not allow it to flow outward from the shaft through fissures or cavities. When ice wall temperatures are lowered appreciably below freezing in order to obtain the desirable increase in strength, the growth in ice wall thickness is of considerable magnitude. In the case of a European shaft recently completed by the freezing method, measurements of the extent of the ice wall revealed a solid mass of ice which was, in places, as much as 70 ft in diam. Measurements of the temperature gradient with depth along the center of the south shaft, indicated rapidly and continuously decreasing earth temperatures to a depth of 112 ft, where the temperature was 32°F. Below 112 ft, the soil temperatures slowly decreased to as low as 24°F. While the recording of actual measurements of the ice wall was impractical at South Shaft, the soil temperatures implied an ice wall condition similar to that of the above mentioned European shaft.

Thawing of the ice wall was allowed to take place without mechanical assistance, that is, brine was not circulated through the freeze holes. This assured slow thawing, as well as minimum soil disturbance resulting from the thermal change in the density of the soil adjacent to the shaft, and prevented severe stresses in the shaft lining. Upon completion of the thaw, the shaft lining had a few needle sized holes which were readily and satisfactorily plugged in less than 2 days time.

Acknowledgment

The author is grateful for the invaluable cooperation and assistance in the preparation of this brief survey of the south shaft project to: Russell G. Haworth, resident manager and James Edmunds, chief mine engineer, of the Potash Co. of America; Frank Harrison, vice-president, Missouri Valley Constructors; Charles W. Lee, vice-president, Consolidated Western Steel, div. U. S. Steel Co.; and V. H. Montgomery, project manager, Winston Bros. Co.

The Paley Report:

Manganese

This is the first in a series of articles on specific aspects of "Resources for Freedom" by the President's Materials Policy Commission.

HIGH-GRADE manganese ore, from which manganese is obtained commercially, is not found in large quantities in any major steel-producing nation in the free world. The U. S. is a "have not" nation with respect to deposits of directly mineable high-grade manganese ore. Known resources of 48 pct Mn or better grade ore amount to less than 200,000 tons.

In 1950 the U. S. steel industry consumed 1.8 million short tons of metallurgical grade manganese ore that contained about 800,000 tons of manganese. About 16 pct of the manganese content was lost in processing, so that about 650,000 tons, or 13 pounds per ton of steel actually entered into steel production. Under present practices use expands directly with steel output, and by 1975 the demand in both the U. S. and the rest of the free world is expected to be roughly 60 pct greater than in 1950.

In peacetime about 80 pct of manganese consumption goes into steel production; high-manganese steel, dry cells, and chemicals account for the remainder.

The manganese supply problem centers around high-grade ore for ferromanganese production. Use of ores containing less than 35 pct Mn sharply increase the costs of making ferromanganese. Use of ferromanganese of grade below 70 pct in turn requires changes in steelmaking that increase steel cost.

Under normal conditions the present small domestic production cannot be expected to increase. Major resources in the U. S. consist of 12 low-grade deposits. The cost of mining and treating these ores to extract a product as good as that yielded by imported ores is at least twice and in some cases more than four times the 1951 price of foreign ores delivered to the U. S.

However, as long as trade relations and overseas shipping are not interrupted, deposits in India, Africa, and Brazil can meet steadily increasing demand at approximately present costs.

Cost considerations indicate that the U. S. should continue to rely upon overseas sources for its peacetime supply, and that this situation is satisfactory. But, this does not take into account the question of how the U. S. will be able to meet its needs in war.

Position of the Rest of the Free World

In 1950, free world steel producers outside the United States, with a steel output of 70 million ingot tons, consumed about 1.3 million tons of metallurgical-grade ore. Their manganese ore demand, expected to increase directly with steel production, will by 1975 be about 2.3 million tons.

Russia possesses over half the known manganese ore reserves of the world and is producing twice the tonnage of any other country. It supplied more than a third of the U. S. manganese requirements up to 1938 and again in 1948, but by 1950 Soviet manganese exports to the free world had virtually ceased. The free world's supply of manganese now comes mainly from India and Africa. Somewhat over 10 pct of U. S. imports came from Brazil and Cuba.

Security Considerations

In the event of war the U. S. might be substantially cut off from 90 pct of present sources.

Reduction in manganese specifications might cut consumption by over 10 pct without seriously affecting steel quality. By elimination of losses in the production of ferromanganese savings as high as 10 pct might be possible. But, wartime manganese requirements cannot be met through conservation alone.

To meet possible future emergencies the U. S. should continue its comprehensive security program for manganese, including stockpiling and research on the economic use of low-grade ore, domestic ores, the recovery of manganese from slag and the reduction of manganese requirements in steel production. If this work, including additional pilot plant operation is pursued vigorously, it should be possible in an emergency to get an adequate supply of manganese from domestic sources. The national stockpile then can be looked upon as a source of supply during the period of at least 2 years required to reach full-scale production from low-grade resources.

Ferromanganese Smelting

In comparison with smelting of pig iron, ferromanganese smelting is a very wasteful process. Under present ferromanganese blast-furnace smelting practice, about 8 pct of the manganese in the furnace charge is lost to the slag, and roughly the same amount is lost to the stack gases; the total loss approaches 15 pct.

Present practice is a compromise between excessive slag loss and excessive stack loss. In fact, it may be seriously questioned whether conventional blast furnace design is suitable for manganese smelting.

U. S. Resources

The known manganese deposits of the U. S. contain a total of 3500 million long tons of raw material and 75 million long tons of metallic manganese. More than 98 pct of this contained metal is in 12 large low-grade deposits of which the most important are those at Chamberlain, S. Dak.; Cuyuna, Minn.; Aroostook County, Maine; and Artillery Peak, Ariz.

Reserves of high-grade ore (48 pct Mn) amount to less than 200,000 tons. About 20 million tons of ore average over 15 pct Mn, and when grade is decreased to 10 pct Mn reserves amount to about 100 million long tons. If cut-off grade is decreased to 5 pct Mn, resources amount to 800 million long tons. Many of these low-grade ores may be beneficiated by flotation or other concentration methods.

Pyrometallurgical Methods

For smelting ferromanganese, it is essential to have an ore containing at least 50 pct manganese, with an Mn:Fe ratio of about 8:1. Direct smelting of 20 pct Mn concentrates is not promising. The only method that offers any promise involves two-step smelting.

Concentrate would be smelted in a blast furnace to produce pig iron and a slag containing over 50 pct Mn. The second step would produce standard 80 pct ferromanganese from this slag.

Four patents by Royster deal with a two-step blast-furnace treatment for low-grade ores. Cost of ferromanganese has been estimated as about $80 above cost of ore used, 10 tons of ore being required per ton of ferromanganese.

Chemical Methods

No appreciable quantity of ferromanganese or metal is produced from domestic ores by chemical processes today, except a small amount produced electrolytically by Electromanganese Corp. at Knoxville, Tenn. Electrolytic recovery of manganese is believed to be limited to applications requiring pure manganese regardless of cost.

However, several of the presently known chemical techniques are almost competitive economically, and large scale plants could be built immediately on the basis of present knowledge if foreign supply were cut off or prices rose.

In 1943 the Manganese Ore Co. opened a plant at the Three Kids, Nev., deposit to produce 300 tons per day of MnO by the sulphur dioxide process. The plant closed after a year because of high costs and foreign ore supply.

(Manganese, Inc., is building a 1200 ton per day plant to treat Three Kids ore, using flotation to produce plus 48 pct Mn concentrate.)

The Chemico process appears to be a practical development of the sulphur dioxide method. Chemical Construction Co. estimates that a plant to produce 60 pct oxide equivalent to 200 tons of manganese per day, treating Cuyuna concentrates, would cost $12 million. Cost of manganese, without credit for byproduct iron, would be $1.23 per long ton unit, compared to current quotations of $1.20 for foreign ores.

Ammonia methods include the Bradley-Fitch and Dean processes. The Dean method appears more feasible. Detailed cost estimates are not available, but Manganese Chemicals, Inc., Minneapolis, Minn. has acquired the Dean patents and is carrying out experimental work.

(DMPA has recently granted a loan to Manganese Chemical Corp. for a 200 ton per day plant.)

Recovery from Waste Products

Annual waste of metallic manganese in open-hearth slags in the U. S. exceeds a million tons; more than our total requirements in the form of ferromanganese. One way to avoid this waste is to recover manganese from slags in the form of ferroalloys and another approach is based on the recovery of manganese from pig iron before it reaches the open hearth.

A three-step, pyrometallurgical method of manganese recovery from flush-off open hearth slags is being evaluated by the Bureau of Mines for the American Iron and Steel Institute in a pilot plant.

The first step utilizes extremely high blast temperatures (up to 3000°F), producing high phosphorous spiegel of 2 to 4 pct P and up to 20 pct Mn. In the second step the spiegel will be partially blown in a Bessemer converter producing high-manganese slag and iron suitable for the basic Bessemer process. The third step is production of standard 80 pct ferromanganese in a blast furnace from the high-manganese slag.

(Recent reports indicate the first two steps are a metallurgical success.)

Based on many assumptions, the manufacturing cost of ferromanganese by the Bureau of Mines process will possibly approach $300 per ton.

The Sylvester-Dean process also is being tried by the Bureau of Mines on flush-off slags.

Plan for Recovering Manganese from Iron

The suggested plan for reducing manganese waste and to expand the economic use of domestic manganiferous ores is based upon two important facts: 1—A manganese content above 0.5 to 0.8 pct in basic iron is not essential to good open-hearth practice; and 2—desiliconization of molten iron, as a preliminary step to its use in the open hearth, speeds up furnace operation and lowers operating costs.

Desiliconization of iron by passing hot metal through an oxidation station would produce a slag which is essentially a manganese silicate. This iron-oxidation slag should lend itself readily to one of several treatments such as the Chemico process.

This suggested method could prove to be a source of more than half our estimated annual manganese requirements, and it would seem logical to augment this source by increasing quantities of native low-manganese ores in blast furnace burden.

An appraisal of the economics of the suggested process, using either the Chemico or the Sylvester-Dean treatment, and crediting the savings in operating cost due to the oxidation step, indicates a cost for standard 80 pct ferromanganese of roughly $100 per long ton, as compared to present price of $185.

Improved technology should tend toward making this country self-sufficient in manganese. Rather than decrease the use of manganese as a final addition to steel, future efforts should be devoted to decreasing the waste of manganese in steelmaking processes.

"As a nation we have always been more interested in sawmills than seedlings"

The Paley Report is important, more important than much that has been said about the economic condition of the United States in many years. It is a story of a country which has used its wealth in great chunks. But, the significance of the report is not in its precis of our industrial past, but what it can foretell.

Undoubtedly it is controversial. Its recommendations are painted on a broad canvas, with sweeping, bold, strokes, and in some cases detailed estimates may be highly questionable. Yet, there is undeniable truth in the large sense of what they have written. That this is inextricably one world, interdependent, related, and a unit that can only be dismembered at the risk of eventual death to all parts.

Many small adversely affected groups—each reacting to the one portion it doesn't like—can give a false impression of overall opposition to the report as a whole. Interestingly, its severest critics have found much to enthuse about.

The realistic approach to costs has hurt the sensibilities of many producers, who envision the end of their enterprises. Their reaction has been personal, and yet understandable. It's tough to be objective, when one sees a threat to the years of sweat, labor and hardship that went into building one's share of industry.

Well written, very readable, but too long, too detailed to be digested in one bite—we feel that the best treatment of the report is to cover various aspects individually. We have chosen to start with manganese, supply, resources, research, and future position; details on specific projects fill in the background.

Eliminating Hand Picking at the Mt. Hope Mine

by Henry Schwellenbach

LABOR shortages, rising wages, and changes in moisture and fines content of the ore necessitated a review of the flowsheet at Warren Foundry & Pipe Co's. Mt. Hope, N. J. Mines Div. This plant had been designed prior to World War II, and started operations in January 1945.

When this operation was designed there was an oversupply of labor in the area, wages were at a reasonable level, the run-of-mine ore was low in moisture, and there was a limited amount of fines. These factors all favored use of hand picking for cleaning rougher lump concentrate.

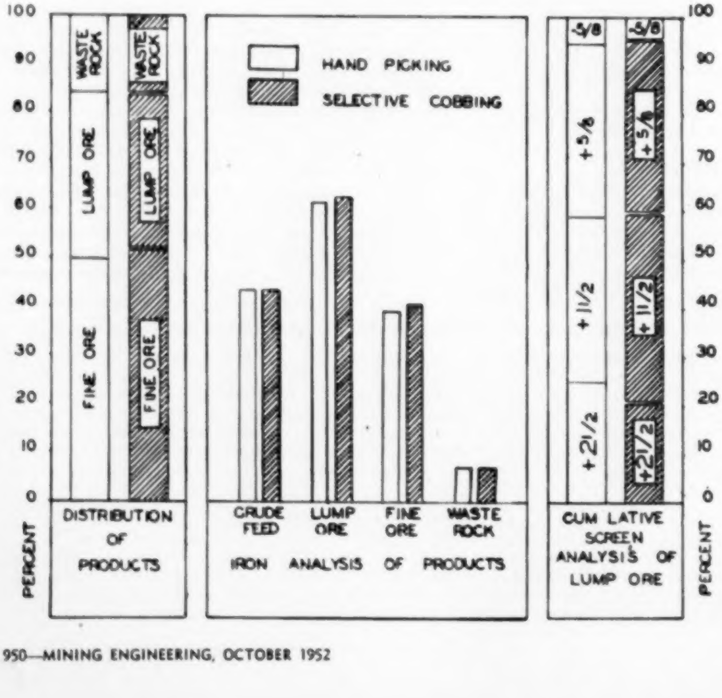
Since that time, many changes have occurred. The proximity of defense industries and large military installations, has created one of the most acute labor shortages in the country. This labor shortage together with present wage levels, and the increase in moisture and fines in the ore created several problems.

The increase in moisture and fines may be attributed to changed mining methods, the extensive use of sprays, and the opening of new mining areas bringing an increase in ground water.

HENRY SCHWELLENBACH is Mill Superintendent, Mt. Hope Mines Div., Warren Foundry & Pipe Corp.

Picking belt is now replaced by selective cobbing.

Two problems were presented by the increase in moisture and fines. The first was that lump ore arrived at the open hearth in an unacceptable condition, due to the segregation enroute of the large amount of fines that had adhered to the lumps. The second problem was that high magnetite content fines coated the large lumps, making it difficult for pickers to distinguish middling by other than the relative weight.



Comparative metallurgical results and screen analyses before and after the change in screens and replacement of picking belt with selective cobber show that the product from cobbing is remarkably similar to that from the picking belt. Part of the change in the amount of $+2\frac{1}{2}$ in. lump ore may be due to change in crusher setting.

Elimination of adhering fines was obtained by screening rougher lump concentrate a second time on a ⅝ in. screen. This eliminated one of the magnetic drums from the circuit with no apparent effect on the grade of that size range. Shipments were satisfactory after this change.

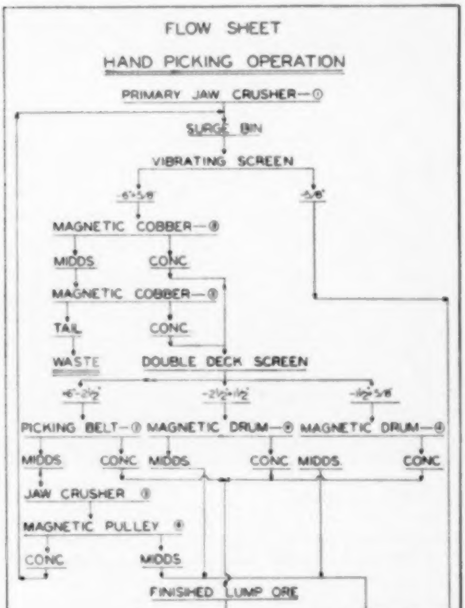
The second problem demanded a somewhat different approach. The questions that arose were numerous. Could a high intensity cobber be selective in its operation on +2 in. magnetite lumps? Would the large pieces, the +4 in. lump, react as did the smaller lumps? Would the upgrading be satisfactory for a lump ore and would the middlings contain any lumps of free magnetite?

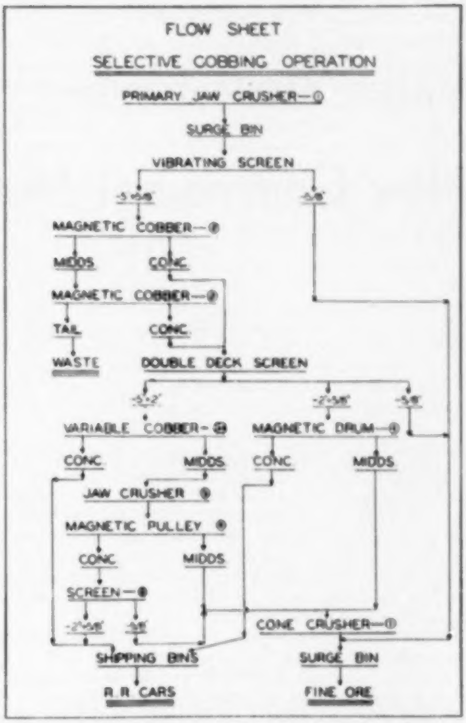
Mill sampling, corroborated by laboratory test work, determined the iron content of the rougher lump to be approximately 54 pct. This figure is approximate because sampling this iron ore at +2½ in. is spotty at best. From these results, it was found that the ore had to be upgraded 6 to 7 pct iron to be acceptable for the open hearth.

Test work on a laboratory cobber was satisfactory, except that +4 in. pieces would not react properly on the small laboratory machine.

In view of the test work findings, a selective cobber was installed, with a variable resistance in the direct current circuit and a variable speed drive on the pulley. The larger pieces reacted properly on this larger pulley. Even so, an attempt is made to keep the primary crusher set at 4 in. instead of the 5½ in. of the former setting.

After one month's operation using different speeds and current intensities, depending on the type and





Before-and-after flowsheets are shown left and above. The inclusion of a second screening operation eliminated one magnetic drum; the addition of the selective cobber eliminated the hand picking belt. Footnotes in the drawings refer to equipment details: 1. 30x36 in., underground. 2. 42x36 in. pulley, 120v, 30a. 105 fpm. 3. 6 to 8 men. 3A. 42x30 in. pulley, 125v, 0 to 30a, 50 to 200 fpm. 4. 36 in. drum, stationary magnets. 5. 13x30 in. 6. 24x18 in. 7. 42 in. Symons. 8. Natural frequency conveyor and screens.

size of ore, the final product proved to be practically equivalent to that from hand picking, see graph.

A third change in the flowsheet may be noted. The magnetic product from the crushing of the selective cobber middling now goes directly to the shipping bins after screening out fines, instead of recirculating. This change, made after sampling showed the product to be of lump ore grade, not only eliminated the recirculating load in the mill, but increased the mill capacity from 110 to 125 tons per hr.

It is impossible to accurately determine the saving with this installation; however, for comparison, the following figures are given. The installation cost was $8,000, which includes $6,000 for the magnetic pulley. The operating cost is infinitesimal, while the manpower requirements of the mill were decreased by 30 pct through the elimination of eight men.

The test work was done by W. M. Aubrey of the Raw Materials Laboratory, Bethlehem Steel Co., and the installation completed by Preston Davenport, assistant mill superintendent. Much of the success of the changes may be attributed to the liberal policy of the management directed by Frank G. Woodruff, plant superintendent.

Enter Wollastonite —

New Commercial Nonmetallic Mineral

by A. L. Hall, R. B. Ladoo, R. N. Secord, and C. A. Stokes

INDUSTRIAL mineral history shows that the entrance of new, nonmetallic minerals into commercial production can be expected to occur from time to time. Latest entrant into the field is wollastonite. Except for one short-lived period in Kern County, Calif., starting in 1933, wollastonite has never before been produced on a commercial scale. But the signs are up for its entrance as an economic factor in the mineral market.

Small scale experimental production from a deposit near Willsboro, N. Y., started about 1943. A pilot plant was built in 1949–50 and irregular, small scale, production has continued. In early 1951 Godfrey L. Cabot, Inc. of Boston, world's largest producer of carbon black, took an operational lease on the property to study the deposits and make market surveys. This became Cabot's first venture in the mineral field, when they took over operations and began design of a new mill, expected in production in 1953.

Properties

The properties and major uses of wollastonite are comparable to New York fibrous talc. It has about the same specific gravity and index of refraction as talc, but is somewhat whiter and softer. The two most important fields of use, paint and ceramics, are the same as for talc.

Ground wollastonite has a fiber length as much as 13 to 15 times the diameter. This particular property makes it applicable in the paint field and others. In addition, the length-diameter ratio may be controlled to rather narrow limits by choice of grinding method. Another important commercial property is brightness or whiteness. Wollastonite reads from 92 to 96 pct of MgO on the G.E. brightness meter for a 99 pct −325 mesh grind.

Purity and uniformity of product are important. Chemically, the wollastonite now being mined analyzes SiO_2 51.76, CaO 47.56, FeO 0.46, MnO 0.04, and H_2O 0.09 pct. Theoretically pure wollastonite analyzes SiO_2 51.75 and CaO 48.25 pct. This analysis indicates over 98 pct calcium silicate, with only small amounts of contaminating ingredients. That these ingredients are not detrimental to color or brightness is shown by the high brightness value of the fine ground material.

A. L. HALL, R. B. LADOO, R. N. SECORD, and C. A. STOKES are associated with Godfrey L. Cabot, Inc.

Willsboro, N. Y. open pit of Cabot Mineral Division shows white face of wollastonite orebody.

By-product calcium silicate from electric furnace production of phosphoric acid has little physical resemblance to this wollastonite, being an off color white, and lacking in fibrous crystalline character.

Minerology

Wollastonite is a calcium metasilicate, $CaSiO_3$, a white, medium hard, fibrous mineral.

In this deposit wollastonite occurs as well crystallized aggregates of bladed crystals intergrown with garnet, and with a few percent of green diopside. The beneficiated product is about 75 pct wollastonite, 15 pct garnet, and 10 pct waste, largely middling.

Wollastonite-garnet ore occurs in at least two beds. The main bed, varying in true thickness from 30 to 70 ft, dips into the hill at an average 30° and is enclosed in gneisses. Diamond drilling has proven the orebody to an average depth of 200 ft along 2000 ft of outcrop with adequate tonnage for many years of quarry operation.

The wollastonite-garnet ratio varies locally, but average composition is uniform over long distances. A friable ore, it is easy to drill, shoot, and crush.

Beneficiation

Unlike talc this ore has to be beneficiated. Successful treatment depends upon the fact that it contains virtually nothing but wollastonite, garnet and a minor percentage of diopside. A very pure wollastonite product can be made by high intensity magnetic and/or gravity separation methods.

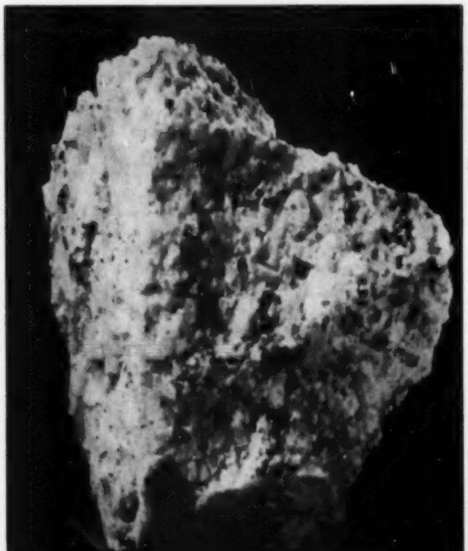
Both garnet and green diopside are magnetic and of higher gravity than wollastonite. Garnet forms a valuable co-product. For many uses the small diopside content of the garnet is not objectionable, but it can be removed by electrostatic separation.

Commercial Grades

Mill No. 1, a pilot plant recently revised for semi-commercial production, has facilities to produce four commercial grades: a C-1 grade for ceramics; P-1 for the paint trade; and two fiber grinds, F-1 and F-2. As information is gained from practical use standards may be modified for these grades and new ones may be added.

Sales experience and research have already (September 1952) indicated the desirability of making additional grades. Among these are a coarser ceramic grade (about 80 pct −325 mesh), a finer paint in the low micron size range, and a special fiber grind for filter aid.

Processing at Mill No. 1 is substantially the same as planned for Mill No. 2. Preliminary crushing in jaw and roll crushers is followed by beneficiation with Exolon magnetic separators and Sutton, Steele and Steele air tables. Fine grinding in conical and tube mills produces the C-1 and P-1 grades. The fiber grinds are produced in a 24 in. double disk Bauer mill.



Crude wollastonite ore. Black grains are green diopside. Cleavage fragment is approximately 2/5 actual size.

A wet process pilot plant recently installed at Mill No. 1 uses an 8 ft x 36 in. Hardinge pebble mill, 18x28 in. Bird centrifuge and settling tanks. This equipment will be used to investigate the production of a P-2 grade to be of finer particle size than the dry ground P-1 grade.

Versatility was the chief design aim for Mill No. 1, used for semi-commercial production of wollastonite and for pilot plant study of a wide variety of minerals and products.

New Commercial Plant

Since Mill No. 2 is to process a new mineral to new products, flexibility of operation and an ability to expand in any direction have been governing design considerations. There will be straight line flow under one roof from ore entrance to rail shipment. An excellent site was obtained, served by the Delaware & Hudson main line, an asphalt road, a 22,000 volt power line, and the Bouquet River. Lake Champlain is within 5 mile truck haul.

The flow scheme incorporates primary crushing and screening followed by drying. Drying insures most efficient beneficiation on magnetic separators and air tables. Garnet and wollastonite are substantially freed at about 10 mesh, but both magnetic and gravity separation give best results on the coarser particles. This necessitates multiple stage crushing in closed circuit to minimize fines. Accurate screening is a must; high capacity separation depends on closely sized feed. Fine grinding is done with a pebble mill-air separator circuit.

Basic process design has evolved from previous pilot plant experience and more recent research by Cabot. Final engineering of Mill No. 2 is in the hands of Charles T. Main, Inc., a Boston engineering firm. The general contractor is Waggaman and Collyer of Glens Falls, N. Y. By September 1952 most of the concrete had been poured, some machinery erected, and steel work started. Prior to the operation of Mill No. 2 early in 1953, demand for all grades will be supplied by Mill No. 1.

Markets

It is believed that wollastonite's uniformity, purity, high reflectivity, and unusual acicularity will contribute greatly to its marketability. In ceramics it has been found possible to produce an ultra low loss electrical body. Wollastonite tiles have been produced which show excellent water adsorption characteristics, high compression strength and good glaze properties. These tiles can be produced in a one-fire process with up to 40 pct fuel saving, and a higher per kiln production rate.

One large firm is using carload lots of wollastonite in producing metalware enamels. They have found that use of calcium silicate in this form has proven more economical than pre-fritting with limestone and silica. Wollastonite in this particular application acts as a natural frit.

Wollastonite must also be considered as a new ceramic material in its own right, not just as a substitute for lime and silica. Research has shown ceramic behavior of bodies containing wollastonite which was unpredictable from the simple molecular combination of lime and silica.

In an entirely different field, wollastonite has been investigated as a paint extender by several leading paint producers. Other uses exploit fiber shape for asphalt tile filler, and in the plastics field.

Several interesting new use developments include a low micron size grind proven of unusual value as an extender for deep-tone flat enamels. An unexpected development has been that of special fiber grinds as replacements for diatomite as filter aids.

As with all new mineral products one expects gradual growth of markets rather than quick and widespread customer acceptance. This new development affords Cabot, with 50 years experience in finely divided pigments, a new and diversified field.

Seismic Survey for Bedrock Depth Determination

Aerial view of the Oso dam site shows approximate location of seismic spreads used for bedrock depth determinations.

by Cleland N. Conwell

APPLICATION of seismic methods to the location of buried channels in placer mining is fairly common knowledge, yet some of the facets of seismic refraction surveys have remained obscure in their application to mining. This description by a member of the U. S. Bureau of Reclamation team investigating bedrock at the Oso Dam site, may offer a lead to wider application of the method for bedrock determinations in engineering of mine and mill plants, tailing disposal, and water supply. It may also offer assistance in planning geological exploration in areas covered by overburden. Pertinent, and perhaps more than passingly significant, is that the seismic survey of the Oso dam site started at noon and was completed the afternoon of the next day.

Location

Oso dam site is on the Navajo River, Archuleta County, Colo., 30 miles southeast of Pagosa Springs, Colo., 10 miles northwest of Chama, N. M. Bedrock at the proposed dam site is Mesa Verde Formation, Cretaceous age sandstone. The nearest good sandstone exposure is about ⅝ of a mile upstream from the location of the seismic determinations. The material above bedrock is sand and gravel with sandy loam soil covering benches above the river.

The site appeared favorable, after a geologic and topographic reconnaissance, but overburden thickness was unknown. Exploratory drilling was not contemplated in the near future so a short seismic investigation was selected as the most suitable method of determining depth to bedrock.

Field Procedure

Geophysically, a seismic depth investigation is simply the measurement of depth from ground surface to the top of a layer which transmits seismic waves at a higher velocity. In this case the high-velocity seismic layer tentatively correlates with the Mesa Verde sandstone.

Field procedure required a seismic exploration line or "spread" with 12 geophones set at measured intervals. Explosive charges were buried at few feet in the ground and detonated at shotpoints near the ends of the spread, and at intermediate points. The time of travel of the seismic waves from the shotpoints through the ground to each of the geophones along the spread was measured and recorded photographically with equipment housed in a field truck.

The seismograph on each spread provided a time-travel graph. From it were computed the speed of seismic waves in various layers below the surface and depth to each different seismic layer below the shotpoint. In the case of the Oso dam site it was possible, in addition, to determine the depths below certain geophones, because there were only two seismic layers involved.

Interpretation

In seismic exploration waves are refracted from the top of layers which often correspond to divisions between subsurface formations. When drill holes are available the geologic logs offer a means of correlating stratigraphic breaks with seismic layers, improving reliability of seismic interpretation in terms of depth to bedrock or other geologic terms.

There were no drill holes available in the case of the work at the Oso dam site. Geologic conditions appeared favorable for seismic technique because of excellent contrast in seismic wave velocity between the overburden and the Mesa Verde formation. The upper layer had a velocity of approximately 2500 fps and was interpreted as overburden, the lower layer had a velocity of approximately 9000 fps and was assumed to be the sandstone. These velocities are of the order expected in these materials.

The seismographs obtained at the Oso dam site were excellent. The energy from the shot came in strongly at each geophone, giving an easily read record. Time-travel graphs plotted from wave arrival times were uniform and made interpretation comparatively easy. In Spreads 1 and 2 near the river the lower velocity layer was 12 to 15 ft deep. For the spreads on higher ground, or the bench south of the river, it was from 15 to 29 ft deep.

CLELAND N. CONWELL has his own geophysical company and is currently employed by the Seattle, Wash., Dept. of Lighting surveying proposed dam sites.

Problems and Trends in Mechanical Loading In Underground Mines in the United States

by Dr. Lewis E. Young

Note: Dr. Lewis E. Young, past president of the AIME, was elected an Honarary Member of the Institution of Mining Engineers this year, and in connection with the award at their annual meeting he presented a paper, extracts from which are given below.

MINING engineers in the United States understand that mining conditions in the British coalfields are much more difficult than in most of the mines now being operated in the United States. We realize also that in the near future we must face some of these same difficult problems.

Early Mechanization

The progress made in mechanical loading in the United States is the result of a long struggle in many coalfields to mine with power tools safely and to increase output per man-shift. Attempts to use power to loosen coal and to transport broken coal in the U. S. may be said to date from the Stanly Header, or Entry-Driver brought from England in 1888. The principle of this boring machine was used in the McKinlay Entry-Driver, almost continuously used in the U. S. since 1920.

Since 1888 experimental loaders, scrapers, and conveyors were installed with more or less success. Beginning in 1918, considerable progress was made with mobile loaders, and in 1920 the first wage agreement for the operation of mechanical loaders was made in Indiana. In 1923 the Pocahontas Fuel Co. loaded nearly 1 million tons of coal using 23 Coloders.

Labor Policy

The United Mine Workers of America have never officially opposed the Mechanization movement. On December 10, 1945, in a statement before the House of Representatives Labor Committee, John L. Lewis said, regarding the policy of the United Mine Workers of America: "We have welcomed progress; we have welcomed machines. We have told our people that they had to accept that condition; that it was the process of progress, and that they would have to take their chances."

Recent Development

In 1951 about 71 pct of the tonnage mined underground was mechanically loaded. Over 4000 shuttle cars are in service and it is estimated that much more than half of the tonnage loaded underground is produced by trackless mining.

In 1947 roof-bolting was introduced extensively and it is estimated that more than 2½ million bolts are now being used per month in about 600 mines. The use of roof bolts has permitted the more effective and safer use of loaders and shuttle-cars.

DR. L. E. YOUNG is a past president of the AIME.

Continuous Mining

The McKinlay Entry-Driver could have been used for continuous mining, but for many years it was used only for entry driving. In 1946 the Silver machine was developed in Colorado, and in 1947 this was acquired by the Joy Mfg. Co. and called the Joy Continuous Miner. Other types of combination mining-loading machines which eliminate drilling and blasting operations are the Marietta Miner, the Colmol, the Lee-Norse Miner, the Junior Miner, the Goodman and the Konnerth machines. There are several other types in the process of development. Probably by January 1953 there will be about 250 continuous miners in operation in the U. S.

Payment of Mine Labor

One of the most important problems in mechanizing was the establishing of rates of pay that would be attractive to the best men. Prior to the installation of mobile loaders the hand loaders and cutters have been paid by the ton. It was felt that a system of a day's pay should be established and that piece and tonnage rates should be abolished completely. Without exception, all coal loaded in mines equipped with mobile loaders is prepared and loaded by men paid an hourly rate.

Trends in Mining Research

There are two diverse approaches in coal production research; to try to design a mining machine to fit current methods, or to adapt mining practice to take advantage of proven machines. A great deal of credit must go to the mine operators who have been enterprising and dynamic enough to use available equipment intensively and to discard it as soon as an improved or new machine is available.

Effect of Changing Markets

Formerly there was an important demand for lump and prepared coal in the larger sizes for domestic use. The use of bituminous coal for house heating has decreased so that not more than 19 pct of the annual tonnage goes to retail trade, only 12 pct is used by the railroads, and much of the retail and most of the railroad coal is in stoker size. As a result of this decline in the market for coarse coal most of the large mining operations have crushers installed in the tipples or preparation plants.

Mass production at the face requires increased preparation facilities. Another important trend is in connection with complete seam mining with mechanical loading. In many mines where the immediate roof is apt to fall with the coal no effort is made to separate roof material from coal prior to

loading. The refuse discarded at some preparation plants is 25 to 30 pct of the tonnage mined.

Summary

1. Competition of coal produced in open-pits and of oil and natural gas has caused owners of deep mines to mechanize. This continuing competition is stimulating development of more efficient mining machines, and mining methods, such as trackless mining and continuous mining.

2. The United Mine Workers of America have not opposed the mechanization program and generally have co-operated fully. The employment of sons of miners in mechanized mines is a promising indication of the acceptance of the mechanization program as the logical way to improve working conditions and the standard of living of the miners.

3. Without exception mines using mobile loaders pay all labor by the day rather than by tonnage.

4. The rapid decline in the markets for lump coal has caused coal operators to pay much more attention to the coal requirements of industry. As a result there has been an increase in complete-seam mining and the preparation and marketing of small sizes of coal.

5. The very extensive use of shuttle cars, trackless mining, and roof bolting has done much to increase the productivity of men and machines.

6. Rapid progress is being made in the development of continuous mining machines and in improved methods of production by their use.

7. In the field of underground transportation there are two major problems, namely, (1) improved methods of taking coal away from continuous mining machines, and (2) long range haulage to mine portals established at strategic locations. Much attention is given both problems.

8. The necessity for mining thinner and deeper seams is causing a serious study of mining methods other than conventional room-and-pillar mining.

9. The availability of capital at low rates of interest has made it desirable to retire inefficient mining machinery and obsolete plants. With the large investment in new mine plants and the continuing cost of ventilating, draining, and maintaining mine workings, consideration is being given to continuous operation and production when there is a market for the output of the mine.

10. Mining research, directly related to production, is now receiving much more attention in the U. S. as it is in Britain, and we trust that we may plan intelligently for "The Mining Industry, Tomorrow and After."

Books for Engineers

Writing the Technical Report, third edition, by J. Raleigh Nelson. McGraw-Hill Book Co. $4.50, 356 pp., 1952.—The aim of the book is to inspire a greater interest in report writing. Basis for the increased interest is a development of a keener understanding of the writer's own mental processes and the practical procedures involved in writing a report. The book is concerned with basic principles and procedures, rather than incidentals. It emphasizes the importance of correct attitudes and suggests a method of approach and organization. In the first part of the book, design and composition are considered. Next, the text gives directions for setting up both the long and short term report. The third part of the book gives a systematic procedure for critical examination of the report. The fourth section presents a 16 week course in report writing.

The Rocks of the Sekondi Series of the Gold Coast, by A .T. Crow. Gold Coast Geological Survey. 7s. 6d, 73 pp., 1952.—The book describes the results of careful and detailed mapping of the Sekondi Series. The Sekondi-Takoradi area is undergoing intensive development, including extensions to Takoradi Harbor and to the railways. Details are given on the nature, extent, and structure of the Series and many faults which cut into them. The book is aimed at those with a practical interest in the area. It is illustrated with geological maps and drawings of horizontal sections. In the back, the terrain of the series is illustrated in a group of plates.

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Industrial Waste Treatment, by Edmund Besselievre. McGraw-Hill Book Co. $7.00, 391 pp., 1952.—The author begins with a discussion of the nature, cause, and general aspects of the industrial waste problem. He establishes basic methods of approach to a solution through the study of causes. Technical subjects covered include sampling and analysis of wastes, pollutional effects, methods of waste treatment, and the recovery of usable materials. Legal factors and economic aspects are covered.

Automatic and Manual Control, edited by A. Tustin. Academic Press, Inc. $10.00, 584 pp., 1952.—The book is a collection of papers from the Conference on Automatic Control organized by the Department of Scientific and Industrial Research, and is intended as reference for those involved with the design of control equipment. The major sections deal with the general theory of servomechanisms, feedback control and related subjects; with process control analysis; the aspects designated as non-linear problems; and with systems working on intermittent data and step-by-step servos. Special sections deal with educational problems, the human operator, and particular applications.

Verdichten Von Leichtbeton Durch Rutteln, by Kurt Walz. Wilhelm Ernst & Sohn. DM 8.00, 27 pp., 1952.—The work is result of laboratory tests on the compaction of light-weight concrete by vibration. The effect of vibration on compressive strength, gross specific weight, and stability (creep strength) of light-weight concrete in relation to vibration rate, amplitude and time is discussed.

Review of Metal Literature, volume 8, edited by Marjorie R. Hyslop. American Society for Metals. $15.00, 884 pp., 1952.—The book is a complete survey of the metallurgical literature published from January to December 1951. The method of subject subdivision is continued in this volume. As in all previous volumes, annotations are provided to indicate the scope and content of each article. An authors index, subject index and table of contents are included.

Ausfuhrung Von Stollenbauten in Neuzeitlicher Technik, by Karl Wiedemann. Wilhelm Ernst & Sohn. DM 18.50.—The book deals with modern methods of tunnel construction. It presents ideas for the experienced engineer who is already familiar with the fundamentals of underground construction work, explosives statics, surveying, and construction machinery. Major sections are on tunneling practice, special machinery and equipment, and operating procedures. Revisions of importance are on drilling, mechanical equipment and roof sealing sections.

Economic Aspects of Sulphuric Acid Manufacture

by William P. Jones

THE consumption of sulphuric acid, one of the most important commodities in our modern industrial world, is often used as a barometer for industrial activity. The economics of acid manufacture are largely dependent upon the location of the place of consumption and the availability of raw materials in that locality.

Sulphuric acid is made from SO_2, oxygen from the air and water. Therefore the sulphur dioxide is the only raw material to be considered in an economic study. SO_2 can be obtained from almost any material containing inorganic sulphur, such as elemental sulphur, pyrites, coal, sour gas and oil, metallurgical gases, waste gases, or gypsum and anhydrite. Many tons of acid can also be reclaimed by the recovery and concentration of spent acids.

The aim of this paper is to present a guide to the economic aspects to be considered when the installation of an acid plant is contemplated. It must be remembered that 1 ton of elemental sulphur produces 3 tons of sulphuric acid and that the shipping of sulphuric acid by tank car is very costly. The size of the plant must also be given careful consideration. For instance, operation of a plant producing 5 tons of acid per day might be warranted in Brazil or Pakistan, whereas economics usually favor buying quantities up to 50 tons per day for use within the United States.

Elemental sulphur, when available at the low price of 1½¢ per lb delivered at an acid plant, has always been the raw material most frequently used for sulphuric acid. All conditions favor its use at this price.

The so-called sulphur shortage has been the subject of so many technical papers, magazine articles, and newspaper items during the past year that it hardly seems necessary to mention it again, but a very brief review of the matter will serve as a foundation for the discussion that follows.

There is no shortage of sulphur. Only a shortage of low-cost Frasch-mined brimstone exists today. Other more expensive sulphur-bearing materials are plentiful, both in the United States and abroad. The low cost of Frasch-mined brimstone has discouraged the development of higher cost sources. However, the approaching depletion of Gulf Coast dome deposits and the greatly increased demand for sulphur here and abroad have made it necessary for industry to prepare for conversion to utilize sulphur in other forms. For future planning this situation must be considered permanent and not temporary. This conclusion is based on the fact that although sulphur demand will continue to rise, the production of Frasch-mined sulphur probably will not increase greatly beyond its present level of about 5,000,000 long tons per year.

The International Materials Conference in Washington estimates 1952 requirements of the free world at nearly 7½ million long tons; and the Defense Production Administration has recently set a new goal for 8,400,000 long tons annual domestic production by 1955. The total sulphur equivalent produced in this country in 1950 was 6 million tons. What, then, are the alternatives for the manufacture of the vital chemical, sulphuric acid?

Today about 85 pct of this country's sulphur, and nearly 50 pct of the world supply, comes from our Gulf Coast salt domes and is extracted from the earth by Frasch's hot water process. The Gulf Coast salt dome deposits have been the most important known natural deposits in the world, producing 90 million tons of sulphur during the past 50 years. However, at the present rate of extraction these deposits cannot be expected to last indefinitely.

Pyrites

Pyrites are, and have been for many years, the source of more than 50 pct of the world's sulphur requirements. The principal use, of course, is in the manufacture of sulphuric acid. The use of pyrites in the United States has diminished greatly because of the availability of low cost native sulphur, but pyrites have continued a major source of sulphur in many other countries.

The most available pyrites for use in this country are in the form of lump pyritic ore and in mill tailings from flotation of other minerals such as lead, zinc, copper, gold, and silver. An important factor, when the use of pyrites for acid manufacture is being considered, is the disposal of calcine. A ton of sulphuric acid requires approximately ¾ ton of high-grade pyrite and results in ½ ton of calcine. If the calcine is a fairly pure oxide, free of harmful impurities, it can be used, after sintering, in an iron blast furnace burden. Its value might be as high as 15¢ per unit of Fe at the blast furnace; or possibly $10.00 per ton of sinter, if it assays 65 pct Fe. This might result in a credit of $4.00 per ton of acid if the sintering plant and blast furnace are both located adjacent to the acid plant. On the other hand, several factors must be considered before this credit can be realized, i.e., freight to blast furnace, availability of sintering facilities, methods of eliminating impurities, and the removal of valuable metal values. In some locations it would be most economical to dump the calcines.

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Lump pyrite and pyrrhotite sometimes run as high as 49 pct S and 45 pct Fe. When this material contains no detrimental impurities it is ideal for the manufacture of sulphuric acid. It must be crushed and ground before it can be burned in a flash or flue-solids roaster, but the resulting calcine, after sintering, brings a premium as blast furnace burden. Lump material is easier to ship and can be stock-piled without appreciable loss due to oxidation.

On the other hand, lump pyrite is usually higher in cost per unit of sulphur, since the selling price must pay for the cost of mining, shipping, and profits. The higher credits for the calcine must be used to offset this additional cost.

The largest readily available source of pyrites in North America is in the form of flotation mill tailings. Most of the copper, lead, zinc, gold, and silver being mined and milled is associated with pyrite. The ore is treated by flotation for the metal values and usually the tailings contain from 20 to 35 pct sulphur in the form of iron pyrite. Because of the low market demand for pyrites during the past several decades, most of these tailings have been stockpiled. Such mill rejects can often be upgraded to 45 to 50 pct S content by another flotation treatment, as pyrite is an easily floated mineral. Capital investment is needed only for flotation equipment, thickeners, dewatering filters, and driers.

Although mill tailings can be purchased at a considerably lower price than lump pyrite and may be equally useful for making acid, the value of the calcines may be greatly decreased for blast furnace burden because of the presence of excess amounts of copper, lead, or zinc.

The capital investment for a pyrite-burning acid plant is approximately twice that for a brimstone plant. Naturally, operating, maintenance, and amortization costs are also higher, owing to the greater amount and complexity of the equipment necessary to produce clean SO_2 from pyrites. Pyrite roasters are required, and an extensive purification train as compared to a simple sulphur combustion chamber for converting brimstone into SO_2.

Pyrites Converted into Sulphur and SO_2

There are a number of places around the world where the supply of pyrites is abundant but where there is no source of elemental sulphur, and there are some uses where it is undesirable to convert sulphur into SO_2 as is done for the manufacture of acid. A process under development now by the Noranda Mines Ltd. of Canada is very interesting in that it converts about 1/3 of the total sulphur, the labile atom, in pyrites into elemental sulphur and the remaining 2/3 into SO_2 which can be converted directly into sulphuric acid. In this way pyrites are converted into three usable products, namely, elemental sulphur, sulphur dioxide, and iron oxide sinter. The successful development and demonstration of this process would be very important to the industrial economy of a number of locations throughout the world.

The greatest waste of sulphur in any form is probably that found in the presence of coal. Not only do huge amounts of sulphur go into the atmosphere from the combustion of coal, but undetermined quantities are deposited on the refuse piles found near coal washers. Many coals contain considerably more than 1.5 pct sulphur, and these must be cleaned before being used for metallurgical purposes. High sulphur coals are to be found across the main coal belt extending from western Pennsylvania through the middle west to Kansas. All through this region coal-cleaning plants are discarding refuse containing 3 to 12 pct sulphur.

To be of value as pyrite, refuse from these plants must be reconcentrated to upgrade the pyrite and reduce the carbon content. The pyrite concentrate should assay not less than 30 pct S and not more than 6 pct C for consideration as acid-plant raw material. Heavy media, tabling or flotation might be used for the concentration of coal brasses, depending upon the characteristics of the washing plant refuse. Coal brasses would be an economical raw material for sulphuric acid manufacture only when the market for the acid is in close proximity to a coal washer capable of producing a large tonnage of low carbon, high sulphur pyrite. The low sulphur values to be found in coal brasses prohibit shipping any great distance.

Spent Oxide

In Great Britain spent oxide from the many gas plants and coke oven plants provides a very significant amount of sulphur. In 1948, the sulphur from nearly 200,000 tons of spent oxide was converted into acid in Great Britain. One of the main phases of England's sulphur conservation program is for the utilization of spent oxide sulphur. However, in the United States the amount of spent oxide available is steadily decreasing to a point where it scarcely becomes a factor in the sulphur picture. The utility companies in New York City have the largest current production of spent oxide in the country. This product runs about 33 pct sulphur, and the total yearly tonnage amounts to only about 1,600 tons of sulphur. It is believed that all the spent oxide on the northeastern seaboard, from Washington to Boston, would total only 25 tons of sulphur per day. Although there are still a number of the larger cities in the West and Midwest that produce spent oxide, the amount available in any one locality is small. Increased use of natural gas throughout the country is causing the gradual decline of this material.

Sour Gases

The sulphur present as an undesirable impurity in many types of industrial gases represents an appreciable potential source of sulphur for sulphuric acid. Exploitation of this source has been started within the past few years. Among the gases which contain available sulphur are sour natural gas, refinery tail gas, manufactured or water gas, and coke oven gas, most of the sulphur being in the form of H_2S. Many problems are posed by the presence of one of the most toxic of gases, hydrogen sulphide, which is corrosive to pumps, pipelines, and other transmission equipment. Generally, therefore, the H_2S must be removed from these gases before they may be utilized, and to a much greater extent than would be economical for sulphur recovery alone.

Consequently, a considerable portion of the work necessary for the recovery of commercial grade sulphur from these varied gases has been accomplished, with the result that the cost of recovering sulphur will not be as great as would be initially imagined. However, the real economy of utilizing the sulphur in sour gases lies in large volume operations. If the source of sour gas is near a market for sulphuric acid, the H_2S can be removed from the gas and burned directly in an acid plant. This results in one of the most economical acid plants. Unfortunately, the source of large volumes is usually

remote from large acid markets, and then the H_2S is partially burned to SO_2, which reacts with the remaining H_2S in the presence of a catalyst to form elemental sulphur.

Metallurgical Gases

The total sulphur emitted to the atmosphere from zinc, lead, copper, and nickel smelters is annually greater than the entire world's native sulphur production. A large quantity of this sulphur can be economically recovered in the form of sulphur or sulphuric acid. However, the big drawback is the location of such smelting operations in relation to the markets for sulphur and acid.

Two companies, Consolidated Mining & Smelting Co. in Canada and American Smelting & Refining Co., have pioneered the research and development of processes for the utilization of sulphur from smelter gases. Consolidated takes 400 tons of sulphur out of 500 million cu ft of waste gases. From these it produces 1,100 tons of sulphuric acid daily, which then is used in the production of fertilizers.[1] All gases from the lead sintering, zinc roasting, and even the acid plants are treated and about 90 pct of the total sulphur values removed and converted to acid.

The flue gases from the combustion of coal and tail gases from a large number of industrial operations contain a small amount, usually only a fraction of a percent, of SO_2, which has always been considered far too weak for economical recovery. An estimated 25 million tons of sulphur go into the atmosphere annually from the combustion of coal alone.

Sulphur dioxide is the most common source of air pollution in the world today. Municipalities and public opinion are becoming more and more conscious of the problem, and eventually utilities may be forced to remove the sulphur compounds from their stack gases. Recent studies[2] indicate that economical recovery of sulphur from such low concentration gases is a distinct possibility, and large quantities of acid could be made at competitive costs.

Gypsum and Anhydrite

Deposits of gypsum, $CaSO_4 \cdot 2H_2O$, and anhydrite, $CaSO_4$, are plentiful throughout the world, and these materials are being used, to a limited extent, as raw materials for the manufacture of sulphuric acid.

The Muller-Kuhne Process was developed in Germany during the first World War, and a commercial plant producing 500 tons of H_2SO_4 per day was later built at Wolfen, Germany, where an ideal combination of raw materials and market for acid and cement existed. In order to make such an operation practical, there must be a saleable lime or cement produced from the calcium in the gypsum. Imperial Chemical Industries have this same combination of circumstances at Billingham in the north of England. Other plants are now under construction for making acid from gypsum in both England and Austria. These developments were undoubtedly brought about by the world shortage of sulphur and sulphuric acid.

The process consists essentially of roasting ground gypsum, or anhydrite, in a rotary kiln, with clay and fuel, such as coke. The basic reaction is

$$CaSO_4 + \tfrac{1}{2}\,C = CaO + SO_2 + \tfrac{1}{2}\,CO_2$$

The clay combines with the CaO to form cement clinker, and the SO_2 gas goes off through a purification system and to a contact acid plant.

Although this process sounds extremely simple, it must be understood that considerable operating and control difficulties are encountered; it is doubtful that the process could be used to advantage in this country where so many other more favorable raw materials exist.

A plant producing 100 tons of acid per day would also produce 100 tons of cement. This is generally considered too small to be economical for a cement plant. Therefore, it seems more logical to build a cement plant of 500-ton capacity, or more, and make H_2SO_4 in quantity required, as a byproduct, from a kiln using gypsum as the raw material. The production of sulphuric acid from gypsum can be regarded as feasible only if cement, or lime, is the main product, the acid being produced as a valuable byproduct. In addition, the plant must be located near the site of the gypsum deposit and the market for the acid must be nearby.

Recovery of Spent Acids

In many cases, sulphuric acid is used in such a way that it is not destroyed and does not go into the product, but it is diluted with water and sometimes other contaminants. Acid used in the manufacture of alcohols, explosives, oils, gasoline, detergents, titanium, and steel, for example, can be recovered by concentration or by decomposition.

The U. S. oil refinery industry ranks high on the list of consumers of sulphuric acid, using about 1.5 million tons per year. This acid does not pass into the product, so that a large percentage is available for recovery. The spent acid is contaminated with petroleum sludge. A refinery must make provision for disposal of its acid sludge, and this is one of the industry's major problems. Some refineries dump this waste product on vacant land, some burn it with other fuel under boilers, and others have installed facilities for recovery and concentration.

Some refinery-spent acids, such as that from alkylation gasoline plants, are high enough in strength and low enough in impurities to be utilized in a number of ways without treatment, provided a suitable market is close to the refinery. This acid at 85 to 90 pct strength can be transported, since it is not viscous and does not attack steel. Alkylation spent acid is being used for the manufacture of fertilizers, and a number of other uses are being found for it. Where no market for such a product exists, alkylation acid must be regenerated into a clean acid. Other spent refinery acids can also be reconcentrated in plants designed especially to deal with various contaminating constituents. The initial cost of such plants is high for small tonnages.

Recovery of sulphuric acid from steel mill pickle liquor is also now feasible, since the cost of new acid is rising. Pickle liquor can readily be concentrated up to 60 pct acid and re-used in pickling baths.

Relative Costs

As previously stated, the largest single factor in any decision regarding raw materials for sulphuric acid is the cost and availability of the materials. Therefore, transportation costs play a very important part in the decision. Table I gives approximate investment and production costs for the various raw materials discussed, at production of 100 tons of H_2SO_4 per day. Both investment and production costs vary greatly per ton capacity with the size of plant. It must be noted that the raw material cost has a great effect on the total cost per ton of acid produced. Table I also shows that acid can be produced more economically by other materials than elemental sulphur, when that raw material is available at little or no cost.

Table I. Comparative Unit Costs Per Daily Ton H_2SO_4 Based on 100 Net Tons Per Day

Raw Material	Capital Investment: Per Daily Ton Capacity, \$	Capital Investment: Factor	Raw Material Cost, \$	Cost Per Net Ton 100 Pct H_2SO_4: Raw Material	Cost Per Net Ton 100 Pct H_2SO_4: Total Cost, \$
Weak acid, 52 pct	3,500	0.6		None	5.60
Sulphur	6,000	1.0	30.00*	9.20	15.10
Weak acid, 30 pct	6,000	1.0		None	10.00
Zn roaster gas	7,500	1.25		None	8.90
Hydrogen sulphide	8,000	1.33	22.00*	6.90	14.90
Alky. spent acid, 87 pct	11,000	1.84		None	12.40
Pyrites	12,000	2.0	20.00*	6.40	19.00†
Refinery sludge, 55 pct	15,000	2.5		None	17.80
Anhydrite	17,500	2.9	2.00**	3.20	25.70

* Per long ton of sulphur equivalent.
** Per ton of anhydrite.
† No credit for calcines.

The \$30.00 sulphur cost is taken as an average cost of sulphur delivered, based on the present \$22.00 f.o.b. Texas Gulf Mines. With this table as a guide, it requires little arithmetic to see that further increases in the price of sulphur will force more and more acid manufacturers to convert or initially construct for the use of other raw materials.

Every raw material listed, except anhydrite, is currently being used in this country for the manufacture or concentration of acid.

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The Cement Industry of Mexico

by Luis Elek

DEVELOPMENT of the cement industry in Mexico began some 40 years ago. It has gradually reached great importance in the economic life of the country and has contributed greatly to the technical and economical advances which have been so noteworthy, particularly during the last 15 years.

From a modest start, the industry now comprises 18 cement-producing plants having a potential annual capacity of approximately 14 million bbl. About half the plants were built and equipped with second-hand machinery during the last world war when no new machinery could be procured. They will hardly attain full production before being modernized and reconditioned, a job now under way in several of the plants.

At present, the yearly consumption is only about 8 to 9 million bbl but is steadily increasing. This is only a fraction of the cement consumed in the U.S.A., which is about 250 million bbl per year, although on the basis of consumption per capita per year, the figure is 1/3 bbl for Mexico and 1 2/3 bbl for the U.S.A. Cement consumption in Mexico is well above the average of all other Latin American countries, with the exception of the Argentine, and considering the potentialities and undeveloped riches of Mexico, the road construction and irrigation works, it is fair to assume that the Mexican cement industry will still prosper and expand in spite of temporary setbacks in the future caused by local events of economic or political nature.

Portland cement is manufactured by burning a mixture of raw materials, one of which is mainly composed of calcium carbonates and the other of aluminum silicates. The most typical materials answering this description are limestone and clay, both of which occur in Mexico in many varieties. To be usable for cement manufacture limestone should contain at least 77 to 80 pct calcium carbonate. Lower grade limestone can be used for cement manufacture only if the lime content is increased by the beneficiation process, which has not yet been adopted by the Mexican cement industry. Most of the limestone used by Mexican cement plants has a calcium carbonate content of 90 to 95 pct and therefore requires addition of one or several correction materials. A typical composition of raw mix for cement manufacture is 85 pct limestone and 15 pct shale, clay or volcanic materials.

Occasionally it is necessary to add a smaller amount of sand or other siliceous material to increase the silica ratio in the raw mix. Likewise, it is sometimes necessary to add 1 or 2 pct iron ore to facilitate fusion in the kiln and to obtain proper relation between alumina and iron oxide. After calcining and

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clinkering in the rotary kilns, the cement produced should have about the following composition:

Feed	Pct
SiO_2	21.5
Al_2O_3	8.1
Fe_2O_3	3.0
CaO	63.6
MgO	2.4
SO_3	1.8

The tendency in Mexico has been to locate the cement plants as near as possible to the consumption centers, such as Mexico City, Guadalajara, and Monterrey. As the limestone deposits, unfortunately, do not always follow the populated areas, it is sometimes necessary to haul the limestone over considerable distances from the quarries to the cement plants. Transport is generally by rail, and since the railroads are inadequate to meet the demands of the industry, severe difficulties are often encountered. Limestone for the cement plant in Guadalajara is brought in over 100 miles of railroad, and precrushed rock for the two cement plants on the outskirts of Mexico City is hauled for a stretch of 40 miles.

All the limestone quarries in Mexico are worked in open pit with drilling and blasting in the conventional manner. Blasted rock is generally handled by power shovel and transported in trucks to the crushing station; a typical installation consists of a pan feeder for uniform feed to the primary crusher, which normally is a jaw or gyratory crusher, followed by a vibrating screen from which the oversize material passes through a secondary crusher, often of the hammermill type. The rock is generally crushed to max 1-in. size and the tendency is to reduce it to ¾ in. or even less to improve the efficiency of the raw grinding mills in the cement plant.

The process of cement manufacture consists of grinding raw materials to form a fine homogeneous mixture, burning the mixture in a kiln to form a clinker, and grinding the clinker with the addition of a small proportion of gypsum to a fine powder. Either the wet or dry process is used, depending on local conditions, the nature of the raw materials, availability of water, and cost of fuel. In general it is considered that a more accurate control of the raw mix composition is possible with the wet process, but the fuel consumption is higher than in the dry process.

Regarding the raw grinding, most of the installations in Mexico are laid out in traditional manner, with preliminary grinding in a ball mill followed by final grinding in a tube mill, both working in open circuit. Lately some plants have adopted more modern closed circuit grinding in comparatively short mills using an airswept circuit and separator, thus obtaining high grinding efficiency and low power consumption, sometimes 2.5 kw-hr per bbl. Usually about 90 pct raw meal passes 200 mesh. These installations refer to plants employing the dry process, for various reasons used almost exclusively in Mexico. Of the 18 cement plants, only 3 are wet process plants, the raw material being ground together with water in ordinary compartment mills until about 90 pct of the slurry passes 200 mesh.

Since the composition of raw materials varies, there is considerable variation in the chemical composition of raw mix leaving the mills. As it is important, however, to obtain a uniform composition of raw mix entering the calcining kilns, a system of homogenization must be installed in connection with the storage silos for the ground material. Homogenization is normally performed by blending silos; sometimes a blending system is used with compressed air, thereby restraining variations of the calcium carbonate titration within very narrow limits.

Calcining and clinkering of the raw mix is performed in rotary kilns. In the 18 cement plants at present operating in Mexico, 46 rotary kilns are installed and 7 more are in the process of being installed. In Mexico the size of the kiln varies greatly, ranging from 6 ft in diam by 60 ft long, with a 24-hr production of 150 bbl, to the largest kiln 11 ft in diam by 350 ft long, with an output of 2400 bbl of clinker. The rotary kilns are fired with heavy fuel oil, or, in northern Mexico, by natural gas. During the passage of material through the kiln calcination takes place, and in the burning zone where the temperature is about 2200° to 2400°F, the chemical reactions between the lime, silica, alumina, and iron take place, resulting in a chemical compound which is Portland cement.

The clinker, which is comprised of small pellets, leaves the kiln and is cooled by air in a clinker cooler to recuperate the heat. This preheated air is afterwards used as secondary air for combustion in the rotary kiln. The fuel oil consumption in the rotary kiln is between 15 to 20 pct of the weight of the produced clinker, depending on the thermal efficiency of the kiln installation.

The clinker is finally ground in mills of the long compartment type, or in short ball mills, from whence it is turned into tube mills for additional grinding with about 3 pct gypsum until approximately 90 to 97 pct passes 200 mesh. Cement grinding in closed circuit with air separation is generally adopted to increase the grinding efficiency and capacity of the mill. The ground cement is stored in silos and ready for packing in paper bags or for distributing in bulk to larger consumers.

The cement is produced according to the official Mexican specifications, which in general are very similar to the ASTM specifications, and it can be stated that the chemical and physical properties of the cement produced in the Mexican cement plants complies with the ASTM specifications for Portland cement in the U.S.A. Recently some Mexican cement plants have also introduced the so-called high early strength cement, which is ground to extreme fineness for the purpose of obtaining the highest possible early compression and tensile strength.

The cement industry in Mexico has shown a tremendous expansion in the last fifteen years and suffers from many shortages due to this quick development. In some plants the power supply is insufficient, in others the railways cannot provide the necessary raw materials or transport the finished cement, and sometimes even the oil supply is limited, with the consequence that the industry in 1951 worked at about 80 pct of its capacity in spite of the ever growing demand for cement. Since low output very seriously increases the cost of production, many of the plants could not show a satisfactory profit. In view of the shortage, the price of the cement was fixed by the government with the usual result of this kind of intervention, a black market giving profits to racketeers rather than to the industry. It is hoped that time will cure these difficulties, and the industrialization program of the government augurs a good prospect for the future of the cement industry in Mexico.

Factors in the Economics of Heat-Treated Taconites

by Will Mitchell, Jr., C. L. Sollenberger, and Ford F. Miskell

Heat treatment of ore prior to comminution reduces power requirements for grinding, reduces grinding media wear, and improves recovery of iron values from a typical Minnesota magnetic taconite. Test data demonstrating this, as well as an analysis of the economics of commercial application of the technique, are presented.

THE taconites in general are hard, tough ores, difficult to grind. Liberation of iron mineral constituents usually is accomplished by grinding the ore through at least 100 mesh, and often it has been found necessary to grind substantially through 325 mesh to achieve satisfactory recovery and grade in the concentration process. Because of the fineness of grind required and the enormous tonnages of material contemplated for treatment in the future, costs resulting from grinding media wear and power consumption, together with capital investment required for comminution, approach astronomical figures. Economy in any one of these elements per ton of material ground could very well reflect a considerable saving to the ferrous industry in yearly costs. With this in mind, the Research Laboratories of Allis-Chalmers Manufacturing Co. have launched a program to discover means of effecting this economy. The initial phase of the work as described here deals with a heat treatment of crushed raw ore, followed by thermal shock in cooling, for the preparation of rod mill feed.

Several investigators[1-5] have noted an improvement in the grindabilities of ores treated in this manner. One investigator[3] subjected low grade iron ores to heat treatment in an electric furnace. Basing his conclusions on screen analyses of crushed products, he observed that treated ore was more easily crushed than the untreated. He reported that observation of the treated and untreated ore through a microscope revealed cracks following the grain boundaries in treated ore, whereas no cracks were present in untreated ore. However, few if any have quantified the improvement in terms of total hp hours saved, reduction of wear of grinding media, or reduction of capital costs of grinding equipment involved. By means of Bond rod mill and ball mill grindabilities[6] and by comparative wear tests, the conditions maintained during this investigation have been definitely evaluated.

Heat treatment experiments have been made on a batch scale in a muffle furnace, followed by continuous scale experiments in a rotary kiln under various conditions of temperature, atmosphere, retention time, and quenching to determine the combination that would give the greatest improvement in the grindability of a taconite without affecting adversely the magnetic susceptibility of the magnetite. From these data, relative costs have been calculated for grinding both untreated and heat-treated ore on a basis of plant capacity of 120 tons per hour. Heat requirements for a rotary kiln, as well as the kiln capacity required for the treatment, have been estimated.

Table I. Chemical Analysis of Lower Cherty Taconite, Erie Mining Co.

SiO_2	48.88
Al_2O_3	0.17
Fe_2O_3	27.79
FeO	15.31
MgO	1.64
CaO	0.94
CO_2	3.07
C (organic)	0.06
P_2O_5	0.067
H_2O (combined)	1.74
Total	99.67
Total Fe	31.32

Table II. Mineralogical Analysis of Ore Sample

Mineral	Wt. Pct
Magnetite	30
Siderite	4
Hematite	5
Silicates	18
Minnesotite	
Greenalite	
Stilpnomelane	
Chert	40

For these tests 50 tons of ore were obtained from the Aurora mine of the Erie Mining Co., Hibbing, Minn. The ore was reported[7] to have been selected from the magnetic portion of the lower chert horizons. An average chemical analysis of this portion is shown in Table I and an approximate mineralogical analysis is shown in Table II.

No attempt was made to differentiate between the various silicates because the compositions vary widely on the different horizons and depend on the degree of oxidation. Minnesotite, however, is the predominant silicate. The siderite grains were dispersed throughout the ore but were essentially associated with the rock-forming minerals.

In a series of preliminary tests in which batches of −6 mesh ore were heat-treated in a muffle fur-

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nace at several temperatures, it was found that slight variations were obtained in the Bond grindability of the samples, determined at 200 mesh. However, it was noted that the circulating loads from the grindabilities became increasingly finer as the treatment temperature was increased. Also the slope of the size analysis curve of the product (Gaudin-Schuhmann plot) became noticeably steeper as this temperature was increased. The same effect was noted with both air and water-quenched products, except that the circulating loads were substantially finer when the treated ore was water-quenched.

It appeared that the grinding characteristics of the ore had changed during the heat treatment, but these changes were not manifested in the grindability value. Observation of the products by means of the petrographic microscope indicated that the grindabilities were being made at a size below the liberation size of many of the constituents of the ore. Fracture across-the-grain appeared to consume an unduly high proportion of the energy. It was obvious that if a conservation were to be made, across-the-grain fractures should be minimized and fracture along grain boundaries increased. A few of the experiments were repeated with grindabilities determined at 100 instead of 200 mesh, and the results substantiated this contention. With increased temperature, an increased grindability at 100 mesh was obtained with the air-quenched product, and an even greater improvement was obtained when the heated ore was water-quenched. Using as a guide the best conditions indicated by the batch experiments, investigators advanced the program to continuous tests in a rotary kiln.

Kiln Tests

The rotary kiln used consists of a cylindrical steel shell in three flanged sections with an overall length of 15 ft 4 in. and a diam of 26 in. The shell is lined with refractory brick, giving an effective inside diam of 16.25 in. It is mounted on a frame with adjustable supports which permit raising or lowering of the shell to any practical slope. The shell is driven by a variable speed motor and is fired with propane gas counter-current to the flow of material. A natural draft system is used. Temperatures can be recorded at 2 ft intervals along the length of the shell by platinum-13 pct rhodium thermocouples imbedded in the refractory lining. An optical pyrometer is used to obtain the temperature of the material being treated as it passes through the hottest zone of the kiln. Fig. 1 is a photograph of this kiln. Feed is introduced 4 in. inside the shell by a 3-in. pipe vibrated with a Syntron mechanism. A retaining ring with an inside diam of 10 in. keeps the feed from overflowing the feed end.

For air quenching, the discharge was collected in the calcine buggy shown in Fig. 1; for water quenching, it was collected in a buggy filled with water.

Retention time was predicted from the formula[6]

$$T = \frac{1.77\sqrt{\theta}\,L\,F}{S\,D\,N}$$

where θ is the angle of repose of the feed (40° for the taconite), L is the length of the shell in feet, S is the slope of the shell in degrees, D is the inside diameter of the shell in feet, N is the speed in revolutions per minute and T is the retention time in minutes. F is a factor required when constrictions are present in the tube. When no constrictions or obstructions are present, F is unity.

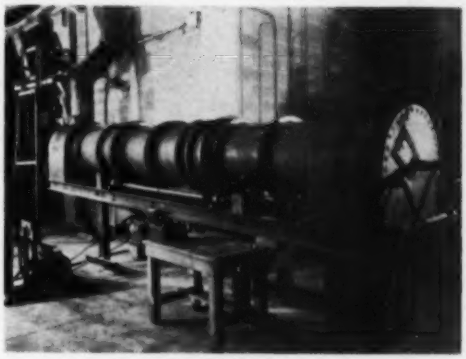

Fig. 1—Rotary kiln, 15 ft x 26 in.

In practice, the volumetric loading of a kiln is usually maintained between 6 and 12.5 pct and is calculated from the formula

$$\text{Loading} = \frac{(CFH)\ (100)}{(60)\ (FM)\ (A)}$$

where CFH is either the feed or product rate in cubic feet per hour, FM is the rate of travel of material through the shell in feet per minute, and A is the cross sectional area of the kiln inside the lining in square feet. In these tests the loading was maintained at 10 pct.

The feed was crushed in a jaw crusher to −3 in., then screened at ¾ in. The oversize was crushed in a Hydrocone crusher and the product, which was about 96 pct passing ¾ in., was combined with the screen undersize. Coke was crushed and screened to −4 +20 mesh for admixing with the taconite when reducing conditions were required.

Two runs were made at 1 hr retention with a hot zone temperature of 1800°F. In the first, no coke was mixed with the taconite, but in the second, 2 pct coke by weight was mixed with the feed. Both air and water-quenched products were collected for each run and rod and ball mill grindability tests made. From the grindabilities, the power required to grind from ¾ in. to 8 mesh in rod mills and from 8 mesh to 100 mesh in ball mills was calculated using Bond's work index formula.[7] The results obtained are shown in Table III.

These results show that practically the same grindability was obtained regardless of the atmosphere used. A considerably higher grindability, however, resulted from water quenching.

To determine whether or not sufficient residual coke remained in the kiln product to cause erroneous

Table III. Effect of Kiln Atmosphere and Type of Quench on Grindability of Product

Atmosphere	Quench	Rod Mill Grindability, 14M G per Rev	Power Requirement, Kw-Hr per Ton	Ball Mill Grindability, 100M G per Rev	Power Requirement, Kw-Hr per Ton
Untreated ore		5.96	2.74	1.15	10.7
Oxidizing	Air	12.1	1.77	1.76	7.38
Oxidizing	Water	24.3	1.15	3.27	6.03
Reducing	Air	12.1	1.77	1.89	6.95
Reducing	Water	22.3	1.22	2.25	6.06

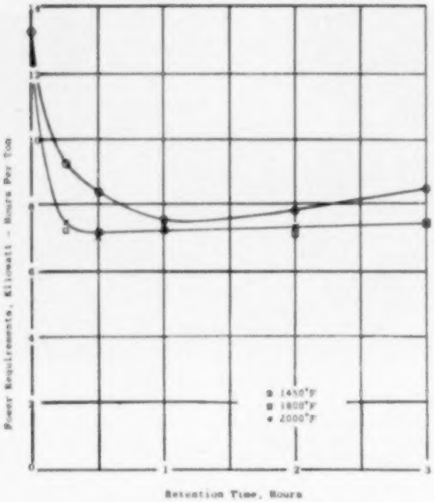

Fig. 2—Grinding power requirements versus retention time at three temperatures.

grindabilities, the −100 mesh grindability product from the run in which coke was used was analyzed for carbon content. It was found to contain only 0.35 pct; obviously any effect on the grindabilities due to carbon would be within the limits of error of the experimental methods used.

A series of runs was made to determine the effect of temperature and retention time on grindability. These were all made with 2 pct coke mixed with the feed, and the products were water-quenched. Runs were made at retention periods of 15, 36, 60, 120, and 180 min. and at burning zone temperatures of 1450°, 1800° and 2000°F as indicated by the optical pyrometer.

From rod and ball mill grindabilities on these products, the total power required to grind from 80 pct passing ¾ in. to 80 pct passing 100 mesh was calculated. The power requirements in kw-hr per ton versus retention time for each of the three temperatures investigated are shown on Fig. 2.

These data show that as the roasting temperature is increased to 1800°F, the power requirement necessary for grinding decreases. Minimum power requirement when a temperature of 1450°F was used was obtained with a 1-hr retention period, and at 1800° and 2000°F with a ½-hr retention period. In all cases power requirements increased slightly with retention periods greater than 1 hr. There appears to be no advantage in using temperatures greater than 1800°F. At 2000°F a black product having a glazed surface was obtained. At temperatures slightly higher than 2000° agglomerates were formed. These latter products were exceedingly friable in the rod mill, but were not so easy to grind in the ball mill as was ore treated at lower temperatures. An examination of Fig. 2 indicates that the greatest overall improvement in grindability was obtained with a burning zone temperature of 1800°F and a retention period in the kiln of ½ hr. It should be explained here that previous experience has demonstrated that the retention period required in a commercial size kiln is double that of the laboratory kiln.

A taconite −2 pct coke mixture was passed through the kiln at a burning zone temperature of 1800°F, and 1 hr retention. The product was water-quenched, then air-dried and passed through the kiln a second time using the same conditions as before, see Table IV. Products were both air and water quenched and the grindabilities determined.

These results show that an additional reduction in grinding power requirements can be obtained by passing the ore through the kiln a second time. However, for the economic comparisons which follow, it was assumed that the treatment of the taconite should consist of a 1-hr pass through a commercial size kiln having a burning zone temperature of 1800°F, followed by water quenching.

Character of Product

The taconite used in the tests contained about 4 pct siderite, $FeCO_3$. One of the advantages of heat treatment was the conversion of the siderite, in a reducing atmosphere, to the magnetic oxide which allowed subsequent recovery in the magnetic concentrate. The −100 mesh ball mill grindability products from the runs at 1450°, 1800°, and 2000°F, all at 1 hr retention, were tested for magnetic susceptibility in a Stearns magnetic tube tester. An increase in the recovery of iron from 86 to 95 pct was obtained, with little effect upon concentrate grade. Maximum recovery was obtained from material treated at a temperature of 1800°F.

Table IV. Effect of a Second Pass

Pass	Atmosphere	Quench	Rod Mill Grindability, 14M G per Rev	Power Requirement, Kw-Hr per Ton	Ball Mill Grindability, 100M G per Rev	Power Requirement, Kw-Hr per Ton
1	Reducing	Water	22.3	1.22	2.25	6.06
2	Reducing	Air	26.5	1.09	2.26	6.04
2	Reducing	Water	31.7	0.97	2.35	5.84

From size analyses of the grindability products as plotted on log-log paper, it was noted that as the grindability increased, the slope of the distribution curve increased. An example of this is shown in Fig. 3. The −100 mesh ball mill product from untreated taconite reported 60 pct passing 200 mesh whereas the heat-treated product reported only 48 pct passing 200 mesh.

This reduction in the amount of fines could well be significant and an advantage, especially in nonmagnetic ores where slimes might inhibit concen-

Fig. 3—Size distribution of untreated and heat-treated grinding mill products.

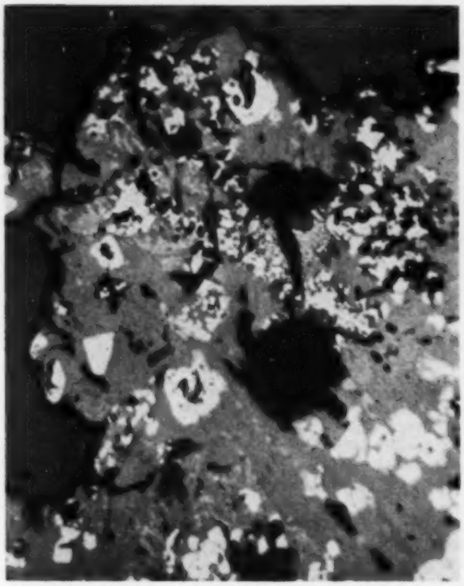

Fig. 4—Photomicrograph of polished section of taconite heated for one hour at 2000°F. Note the coalescense of the magnetite crystals. X250.

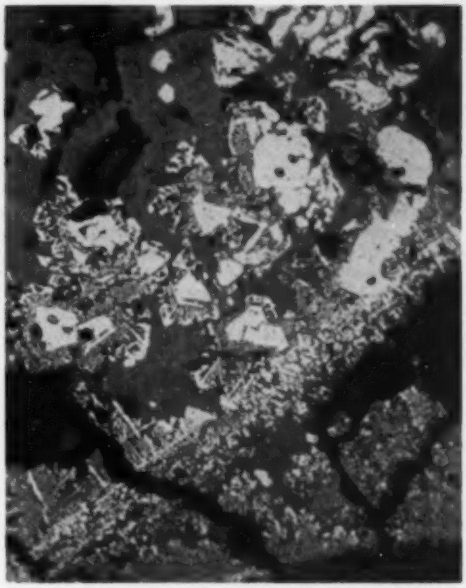

Fig. 5—The same specimen as shown in Fig. 4 illustrates the dendritic growth of magnetite crystals and the coalescense of these dendrites into larger euhedral crystals. X500.

tration. However, it is possible that reduction in fines content might have deleterious effects on pelletization properties of the magnetite concentrates.

Six thin sections and 14 polished sections of untreated feed and heat-treated products were examined microscopically for evidence of recrystallization or cracking which might account for the increases in grindability. No appreciable differences in grain size or crystal development were noted in any of the specimens treated for 1 hr or less below 1800°F, although there was slight evidence of bridging of magnetite grains in specimens treated for 2 hr at 1450°F. In specimens treated at 2000°F, there was abundant evidence of recyrstallization and coalescence of the magnetite. The magnetite had recrystallized into fine dendritic groups. Upon further growth, these dendrites coalesced and formed large euhedral crystals. Fig. 4 shows coalescence of anhedral to subhedral magnetite grains. Fig. 5 shows the dendritic structure as well as coalescence.

A thorough examination of the thin sections resulted in one possible explanation for the increase in grindability. The untreated feed contained siderite dispersed throughout the ore, whereas the heat-treated samples showed an absence of siderite. During the heat treatment, the siderite had been changed to iron oxide. With the chemical change there would be an acompanying decrease in volume from the original space occupied by the siderite to the space occupied by the oxide. The voids thus produced would tend to weaken the structure and increase the grindability. The increase in grindability can also be attributed to the creation of cracks or zones of weakness through differential thermal expansion of the various components of the ore during heating and differential contraction during the rapid cooling afforded by the water quench.

Heat treatment may be accomplished by several means such as the vertical shaft furnace, hearth roaster, sintering machine, fluosolids furnace, or rotary kiln.

From the test work reported here it can be seen that the substantial increase in grindability is, in the main, dependent upon the thermal shock resulting from a water quench of relatively high temperature products; hence a heat recuperating system is practically precluded.

The thermal efficiency of the vertical shaft furnace has been demonstrated by Fred D. DeVaney[10] but it should be noted that the treated product is discharged from the shaft at a temperature in the neighborhood of 300°F. A water quench applied at this temperature would probably not produce sufficient shock to a magnetic taconite. An improved grindability would not be realized, especially where no chemical change or little recrystallization is effected by the heat treatment.

Besides the inherent economic operating disadvantages of a hearth roaster, no thermal conservation over that enjoyed by kiln processing is obvious at this time.

The sintering machine, as well as the fluosolids furnace, operates most effectively on a sized feed, although in combination, one could handle rod mill feed with the fines removed, and the other, ball mill feed. This would assume a screening operation at perhaps 8 to 10 mesh of the raw rod mill feed. Again, a heat recuperating system could not very well be applied in the use of this apparatus for the reasons given above; hence the Btu content of the discharged product, accounting for a large proportion of the heat loss, would be similar to that of a rotary kiln. In view of these factors, the cost of heat treatment was estimated only for a rotary kiln plant.

To heat taconite 1 hr in a kiln having a burning zone temperature of 1800°F, at the rate of 120 tons per hour, an 11-ft 6-in. by 300-ft kiln would be required. Total equipment costs for a kiln plant including drive, controls, burning equipment, exhaust system, conveying equipment, and water quencher would be approximately $451,200.* About 700 installed hp would be required to operate this plant.

* Based on 1951 U.S. dollar.

To heat one ton of taconite, having an assumed specific heat of 0.3, from 70°F to 1800°F, 1,040,000 Btu's are required. If it is assumed that the radiation loss will be 20 pct of the above figure and 10 pct moisture must be evaporated from the ore, a total of 1,548,000 Btu's are required. When one kiln is used to heat this tonnage of ore to 1800°F, the exit gas temperature may be as high as 1000°F, and fuel efficiency will be about 59 pct.[11] Total heat requirements then are 2,620,000 Btu's per short ton. If bituminous coal having a heat value of 13,800 Btu's per lb is used as fuel, at $0.30 per million Btu's, fuel costs per ton of taconite treated will be $0.79. A kiln plant estimate is shown on Table V.

Economies Effected

As previously noted, one of the advantages of heat treatment of the taconite was an increase in recovery of iron. The ore used in these experiments contained only about 4 pct siderite. For magnetite ores high in iron carbonates a much greater improvement in total iron recovery would be obtained.

Total power required to grind the untreated ore from 80 pct passing ¾ in. to 80 pct passing 8 mesh in rod mills and on down to 80 pct passing 100 mesh in ball mills was calculated from the grindability results and Bond's work index to be 13.44 kw-hr per ton. The power to grind heat-treated ore with the same reduction was 7.44 kw-hr per ton or a saving of 6.00 kw-hr per ton.

To determine the relative abrasiveness of heat-treated and untreated taconite, samples of each were passed through a hammer mill operating at 1800 rpm with two hammers and with grate bars spaced at ⅛ in. Approximately 90 lb of untreated feed were put through the mill to determine the feed rate which would properly load the mill and case harden the wearing surfaces of the hammers and grate bars. Then the samples listed on Table VI were passed through the mill. The hammers and grates were weighed before and after each test. The data obtained are shown on Table VI.

Table V. Estimated Operating Expenses for Kiln Plant Heating 120 Tons Per Hr at 1800°F for Retention Period of 1 Hr

Item	$ Per Ton
Power, estimated available at $0.011 per kw-hr	$0.048
Labor, 4 1/3 men per shift, $1.50 per hr avg	0.0541
Maintenance, 10 pct equipment cost annually, 330 days per year	0.0475
Write-off on installation,** taxes, depreciation at 20 pct of equipment cost annually	0.238
Fuel	0.79
Total	1.1776

** Assumed to be 2.5 times the equipment cost.

Table VI. Pulverator Wear Tests

Item	Untreated	Treated
Total weight of feed, lb	597	423
Feed rate, tons per hr	2.56	3.06
Avg kw demand	12.72	6.80
Power requirement, kw-hr per ton	4.96	2.22
Total wear on grates and hammers, lb per ton	1.097	0.448
Total wear in lb per hp-hr	0.254	0.151

Table VII. Relative Grinding Plant Costs, 120 Tons Per Hr

Item	Cost Per Ton: Untreated Ore	Cost Per Ton: Heat-treated Ore
Power	0.148	0.082
Labor	0.021	0.021
Maintenance	0.034	0.021
Write-off	0.182	0.106
Ball and rod maintenance	0.189	0.062
Liner maintenance	0.019	0.006
Total	0.593	0.297

One rule of thumb states that ball and rod wear are directly proportional to the horsepower hours per ton required to grind an ore and that this wear is about 0.15 lb of metal per horsepower hour for magnetic taconite. This same rule states that liner wear is roughly 10 pct of the ball and rod wear. Using this rule and the same ratio of wear for untreated and treated ore obtained in the hammer mill tests, it was calculated that for untreated ore, ball and rod wear is 2.7 lb per ton and liner wear is 0.27 lb per ton. For treated ore, ball and rod wear is 0.88 lb per ton and liner wear is 0.088 lb per ton. By heat-treating the ore, a wear saving of 1.78 lb of metal per ton of ore milled should be possible.

For a basis of comparison of capital equipment costs, a convenient tonnage was 120 tons per hr or 2880 short tons per day. To grind this tonnage of untreated ore, one 10½x12-ft overflow rod mill and two 10½x12-ft overflow ball mills would be required. To grind heat-treated ore, one 9x12-ft overflow rod mill and one 10½x14-ft overflow ball mill would be required.

The omission of the one mill would probably make little difference in the number of men required to operate the grinding circuit with the tonnage used as a basis. Consequently labor costs should be essentially the same.

For a comparison of grinding plant costs, each plant handling 120 short tons per hr, the following assumptions were used. Metal replacement cost was assumed to be $0.07 per lb and power was assumed to be available at $0.011 per kw-hr. Labor requirements were assumed to be 1 2/3 men per shift at an average hourly rate of $1.50. Annual maintenance costs were assumed to be 10 pct of the equipment costs and the total installation was assumed to be 2.5 times the equipment cost. Write-off on the installation, taxes, and depreciation were assumed to be 20 pct of the equipment cost annually. A 330-day year was used. Total equipment costs for grinding untreated ore were estimated at $326,897. For grinding treated ore these costs were estimated at $200,119. The relative costs for grinding treated and untreated ore are shown on Table VII. Table VII shows that heat-treating the taconite ore makes possible a saving of $0.296 per ton in grinding costs.

Conclusion

At the present time, with the conventional apparatus available today for pyroprocessing, heat costs generally are high in comparison to power costs in comminution. It was found that on a basis of plant capacity of 120 tons per hr the saving in overall grinding costs would be $0.30 per ton,

whereas heat treatment in a rotary kiln installation would cost about $1.18 per ton.

The heat treatment of ores in general solely to reduce grinding costs probably cannot be justified on that basis alone at the present time. However, this technique does furnish the additional advantage of reducing grinding media wear, and in this particular case improves the recovery of iron brought about by the conversion of siderite to magnetite. The saving attained through increased iron recovery has not been included in the economic comparisons.

The substantial reduction in the percentage of slimes produced during grinding of heat-treated ore could enhance the applicability of the flotation process in nonferrous beneficiation, and coupled with the other advantages might justify the pyroprocessing.

Acknowledgments

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Technical Note

The Tromp Heavy Media Process

by John Griffen

THE distinguishing principle of the Tromp process is the use of a medium in the bath which is not stable, i.e., the solids will settle and the density of the medium increases with depth. A medium of uniform density can most easily be effected by agitation or by means which increase the viscosity of the medium. Either of these works against the sharpness of separation of coal from its impurities.

The Tromp process grinds the medium solids to relative coarseness, allows them to settle downward in the bath without hindrance, and introduces horizontal currents of medium at intermediate levels between top and bottom of the bath. This prevents any accumulation of particles denser than the upper layer of the medium from which the clean coal is conveyed away or an accumulation of particles lighter than the bottom layer or medium whence the refuse is removed. In a relatively shallow bath making a two-product separation, the intermediate horizontal currents sweep these intermediate gravity materials away to be discharged with the refuse removed by the conveyor. Such a system provides a number of advantages.

The gravity of separation can be changed simply by changing the density of the medium without changing particle size of medium solids or other conditions, as would be necessary with a system requiring uniform or stable density throughout the bath. Some installations in Europe separate at 1.40 sp gr on one shift, producing a superclean coal, and on the next shift reclean refuse from dry cleaning units and effect a separation at 1.80 sp gr.

Medium solids may be much coarser in size consist, which reduces the viscosity of the medium and promotes its fluidity and in turn the sharpness of separation. Coarser medium solids with a more rapid settling rate are much more easily recoverable from the dilute medium rinsed from the products.

The Tromp process is not limited to the use of medium solids having magnetic properties. Instal-

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The subject of this paper, TN 131F, is discussed by the author at greater length in a paper scheduled for publication in the Blast Furnace, Coke Oven and Raw Materials Proceedings, AIME, 1952, Vol. 11. Manuscript, March 13, 1952.

lations in Europe use a variety of solids which have the required specific gravity to produce the desired bath densities having the desired fluidity. Magnetite is being used in some instances.

A particular method of recovery and reconditioning of the medium solids from the rinsings is not peculiar to the Tromp process. In its early development, some fifteen years ago, gravimetric methods of recovery were first used, even though a magnetite medium was employed. Magnetite has the advantage of being readily recoverable from the rinsings by fairly simple means, consisting of screening at 60 to 70 mesh to remove the coarser shale and coal impurities and afterward classifying in a simple conical tank to remove the slimes contamination from the raw coal treated.

Magnetic recovery and reconditioning can be used if magnetite or other magnetic solids are utilized. However, these operations, as employed in commercial enterprises, have been so successfully accomplished by the combination of screens and hydraulic classification that they seem preferable to magnetic methods.

As carried out in connection with the Tromp process gravimetric recovery offers some advantages, particularly when separations are made below 1.8 sp gr. With the bath medium at densities lower than this figure, the average density of the medium solids can be well below that of reasonably pure magnetite and yet have a relatively low percentage of solids by volume which will assure a fluid medium. For example, with separation at 1.45 sp gr, if the average density of the medium solids is 3.25, the percentage of solids by volume will be only 20 pct and the viscosity of the medium will be exceedingly low. These medium solids would contain only about 40 pct by volume of magnetite, sp gr 5.0, the balance being mainly coarser shale contaminating solids.

The amount of medium which must be removed from the bath is a function of the contaminating solids entering the bath with the feed. It is readily seen from the above paragraph that, with a given contamination, the amount of magnetite which must be handled in the recovery system is reduced when the solids in the rinsings contain only 40 pct by volume of magnetite.

Furthermore, a high recovery of magnetite is promoted by the fact that a high proportion of the recovered solids are the coarser portion of the contaminating solids. Tests on the solids in recovered medium show that the coarsest sizes consist of shale contaminated with a little coal while the intermediate sizes are largely shale and the finest sizes are magnetite with some shale. The following is an example:

Microns	Wt, Pct	Solids, Sp Gr
150	2.0	1.84
150 to 120	4.0	1.98
120 to 88	12.8	2.60
88 to 0	81.2	4.06
Total	100.0	3.57

The classification effected by the gravimetric recovery system makes the contaminating solids recovered and re-used as medium fairly coarse, which in turn reduces the viscosity of the medium. Experience has shown that these coarser shale particles do not build up in the recovered medium. This is seemingly due to their being less hard than magnetite so that they are degraded more rapidly than the magnetite and are continuously eliminated by the hydraulic classification in the recovery cone.

The Tromp process has been designed with automatic controls on the essential operations so that operator attention can be reduced to a minimum. An automatic density regulator controls the density of the recovered medium to that required for the separating density desired in the bath. The setting of this regulator can be readily changed.

To compensate for another variable which arises from variations in moisture content of the raw feed coal due to inevitable differences in size consist, a bath automatic regulator is also provided. Variations in feed moisture will dilute the bath medium and in turn the medium drained from the products and returned directly to the bath. The bath regulator measures the density of the returned medium, and when it drops below the desired figure diverts the required proportion to an auxiliary small settling cone which densifies the diverted stream to compensate for the water added with the feed.

Correction

In the September 1952 issue: TP 3382BH. Glass and Chemical Sand Manufacture in the Edwards Paddle Scrubber, by Will Mitchell, Jr., T. G. Kirkland, and R. C. Edwards. P. 875, col. 2, should read: Will Mitchell, Jr. and T. G. Kirkland, Members AIME, are with Research Laboratories, Allis-Chalmers Manufacturing Co., Milwaukee. R. C. Edwards is with the Processing Machinery Department. P. 876, par. 2, line 2, should read "feed rate of slurry." P. 876, par. 4 should read:

If the screening had been done wet, the sand was dewatered to 12 to 18 pct moisture by means of a rake classifier where the −200 mesh fraction was eliminated. When the feed was dry screened, it was necessary to add water by means of a spray at the feed end of the scrubber and the fines were allowed to pass through the scrubber to be removed later in the final classifier.

P. 879, Table VII, col. 3: line 3 should read 0.973; line 6 should read 1.924; line 12 should read 1.268; line 15 should read 3.190. P. 879, Table IX, col. 5, line 8, should read 6.3.

Professional Training of the Geophysical Engineer

by James B. Macelwane, S.J.

Historically whenever application of scientific results to a new problem required the special experimental background, the economic outlook and the practical knowledge characteristic of the engineer, a curriculum in that type of engineering based on the professional requirement grew up in the engineering colleges. This is precisely what is happening now in geophysical engineering.

THE Mineral Industry Education Division of AIME has long been interested in geophysical education. Organized in 1938, a Committee on Geophysical Education presented a number of reports in succeeding years. The AIME is an engineering society; hence it might be expected that the interest of the Mineral Industry Education Division would have centered about the relation of courses in geophysics to engineering education. However, this was not the case. The burden of the discussion held under the auspices of the Committee on Geophysical Education bore on geophysics in general and particularly on its relation to geology.

In the decade that has passed since those meetings much water has flowed under the bridge. Not only have the geophysical sciences developed at an extraordinarily rapid rate but the practical importance and the range of their applications have spread in ever widening circles and have even assumed a very prominent place in research programs of the military, the Army, the Navy and, particularly, the Geophysical Research Directorate of the Air Force at Cambridge, Mass. So enormously have the geophysical sciences as such grown in scope and in degree of specialization that no one man in a lifetime of study can hope to master the theoretical and experimental complexities or even to follow intelligently in the published literature all the varied accomplishments of basic research in these fields. As in the other sciences, most of the results of geophysical research have potential economic value and are being applied to a wider and wider range of human needs.

Here we see a parallel with the older sciences and their relation to engineering. Whenever the applications of scientific results required the special experimental background, the economic outlook, and the practical knowledge characteristic of the engineer, a curriculum in that type of engineering grew up in the colleges and universities. In the words of the "Report of the Committee on Adequacy and Standards of Engineering Education" in the January 1952 number of the *Journal of Engineering Education:* "Engineering education in America was not developed after a preconceived plan. It evolved in parallel with the needs of a growing country, a country engaged in the development of a vast industrialization. From the first it filled a practical need, and its design emanated largely from an evaluation of the professional requirement." The advent of the railroad created a need for scientifically trained engineers and the response was the curriculum in civil engineering. The development of steam power created the demand for a curriculum in mechanical engineering. Neither of these types of engineers was trained to cope with the requirements of electrical power, and so there came to be a curriculum in electrical engineering. The advent of radio communication and of electronics during the first World War and in the years that followed resulted in the establishment of electronics engineering, sometimes within the curriculum in electrical engineering and sometimes independent of it. The complexities of industrial management have created an ever increasing demand for industrial engineers and the consequent building up of a curriculum in industrial engineering. The wide industrial application of chemical processes required a type of engineer whose background was training in chemistry but who was able to design processes and plants on an industrial scale; hence we have the curriculum in chemical engineering. The advent of the airplane created an entirely new set of problems which were solved by the training of aeronautical engineers. Many of these have called for more and more science and mathematics. Such was the history of the older engineer-

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Table I. Distribution of Subjects in a Curriculum* Leading to B.Sc. Degree in Geophysical Engineering, Saint Louis University, Accredited by ECPD

Humanities	Mathematics	Science	Engineering
Freshman English	Engineering mathematics	Freshman physics	Technical drawing
Report writing	Calculus (3 semesters)	Engineering physics	Descriptive geometry
Speech	Differential equations	Chemistry	Surveying
Philosophy	Vector analysis	Physical geology	Plane-table mapping
History	Potential theory	Historical geology	Machine tools
Business management	Managerial accounting	Minerals and rocks	Applied mechanics
		Structural geology	Thermodynamics
		Engineering astronomy	Hydraulics
			Electrical engineering
			Electronic circuits
			Strength of materials
			Structural and ground vibrations
			Force fields of the earth
			Exploration geophysics

* Physical education is included in this program.

ing curricula and such too is the history of geophysical engineering.

What is a geophysical engineer? What is an engineer in general? There would seem to be three characteristics of the professional engineer on which all can agree. In the first place, an engineer must be equipped with a store of fundamental scientific knowledge sufficient to enable him to solve the problems inexorably thrust upon him in the practice of his profession. In the second place, he must have acquired the technical skills required for planning and design of structures, apparatus, circuits, processes, or procedures in his field that will give the optimum performance at the least cost, and he must be capable of making a reliable estimate both of the performance and of the cost so as to obtain the most for the outlay that is justified by the foreseeable overall returns. Thirdly, he must be a capable manager and must know how to deal with men. These three general characteristics give at least a partial definition of a geophysical engineer. In the large and important class of geophysical problems that arise in prospecting for petroleum and other minerals, the geophysical engineer is immediately faced with the task of planning a program, and in his planning he must solve two problems: 1—He must decide what is the most effective procedure that can be applied in the given circumstances. 2—He must weigh the performance of the various methods against the economic limitations of the situation to secure the maximum of useful information at the lowest justifiable cost. Obviously, both the planning and the execution of such a project will require a thorough training in the geophysical sciences, and unless the geophysical engineer has this solid background of geophysical science, he cannot choose intelligently either the sciences to be applied or the method of their application. Without this solid background of geophysical science he will neither know the limitations of the method itself nor realize the pitfalls of instrumentation and interpretation. It is by no means enough for a geophysical engineer to know physics, geology, and electronics engineering. He must know geophysics. On the other hand, he must not be merely a scientist, however competent; he must be above all an engineer. He is applying physical principles verified in the laboratory to an uncontrolled medium, the earth. Unless he is able to superimpose on his thorough knowledge of geophysical science the practical economic outlook of the engineer, he cannot forecast intelligently either performance or cost.

Geophysical exploration is only one of the many fields that call for the special abilities and experience of the geophysical engineer. The Seismological Engineering Foundation on the Pacific Coast came into being because the problems which antecede and underlie the design of structures to resist the destructive action of earthquake motions transcend the potentialities of the civil and structural engineer. On the other hand they have too much of an engineering aspect to interest the geophysical scientists. The problems are seismological, but they call for the background and practical knowledge of geophysical engineers.

A further broad field into which geophysical engineering has expanded in the last few years is a study of the impact of vibrations on buildings and other structures, particularly in urban communities, caused by quarry and mine blasts, by trip hammer blows, by pile driving, by vibrating machinery, and by highway and railway traffic.

Again, in the quarrying and strip mining industry, the skills of an engineer are needed for the general application of geophysical methods in determining overburden and in mapping the topography of a buried rock surface. The services of a highly trained geophysical engineer are required in the foundation work connected with the construction of highways, dams, bridges, and similar structures, and he is called upon to deal with the closely related problem of locating water supplies and developing water resources by geophysical means. Most of these activities are illustrated by papers on the geophysical program of the AIME annual meeting, February 1952.

In view of these circumstances, what conclusions must be made concerning requirements for professional training of the geophysical engineer? As in all other branches of engineering education, the ideal curriculum will be expected to produce two results, professional attitudes and professional competence. The professional attitude of an engineer is that of a public servant who is conscious of his social responsibility not only to his client but to public welfare in general. Professional competence requires that he have the knowledge and skill to utilize geophysical science in an economic manner for the practical solution of any problem that is brought to him. His training, therefore, as shown in Table I, will be in four main areas. The first of these is the field of humanities. This type of training is the foundation of clear and forceful thinking, the ability to analyze a problem, to organize the subject matter of a report in correct perspective and to write a report in a telling and attractive style. The second area is mathematics. The third comprises the fundamental sciences, particularly physics, geophysics, and geology. The fourth, engineering, includes both general training and specialized geophysical engineering.

Titanium Dioxide Analysis of MacIntyre Ore by Specific Gravity

by Alan Stanley

THE MacIntyre Development of National Lead Co. is located at Tahawus, N. Y., in the heart of the Adirondack Mountains. Operations involve the mining and concentrating of a titaniferous iron ore to produce ilmenite and magnetite concentrates. A general description of the operation and metallurgy has been given by Frank R. Milliken.[1]

Pigment plant production demands that the MacIntyre mill produce a 44.7 pct TiO_2 ilmenite concentrate. To achieve the required ilmenite grade and tonnage it is important that the table concentrate grade be closely controlled. Unfortunately, however, the titaniferous orebody which feeds the MacIntyre mill is not uniform. Ore dressing characteristics vary from one end of the orebody to the other, and from one level to the next. The changeable nature of the mill feed precludes a single adjustment of the equipment for long periods of time. Thus the operators must constantly watch the equipment to insure a uniform concentrate from the fine and coarse tables and Wetherills, or dry magnetic separators.

Chemical assaying of mill products requires about 4 hr from the time the sample is taken until assay results are obtained, and this is available only on a two-shift basis. The ore may change rapidly, even several times during a shift, so that assay results lose most of their control value by the time they are reported to the mill operating crew. Members of the crew have therefore tried to evaluate the table and Wetherill concentrate by visual inspection, since through long experience the shift operators, under most circumstances, can gage closely the grade of the mill products. However, there are times when the physical nature of the ore is radically different from normal. Under these conditions visual inspection is of no value, and at such times final ilmenite as low as 43 pct TiO_2 has been produced and shipped before the assay results have been received. The specific gravity method of assaying for TiO_2 has been attempted to eliminate the shipping of ilmenite below normal grade as well as to help control day to day and hour to hour mill production.

Table I shows the minerals found in the MacIntyre ore along with their average weight proportions and specific gravities. The first two products considered for the specific gravity method were fine and coarse table concentrates. It was reasoned that these products were essentially ilmenite with the higher specific gravity gangue minerals. Since they were always produced the same way, and the desired grade of TiO_2 was always constant, the specific gravity of these materials would increase or decrease as the amount of ilmenite increased or decreased. Thus for table concentrates which assayed 40 pct TiO_2 a constant gravity would invariably be obtained, and as the TiO_2 value changed the specific gravity would change in direct proportion.

The third product considered was Wetherill ilmenite. It was assumed that a desired grade of 44.7 pct TiO_2 would also always contain the same amount and type of gangue minerals along with the ilmenite, and thus would always have the same specific gravity. As the TiO_2 value of the ilmenite concentrate changed so would its specific gravity.

Dr. Kenneth Vincent, chief metallurgist of the Baroid Division of National Lead Co. at Magnet Grove, Ark., ran specific gravity tests on 17 samples of the desired products. The lowest specific gravity reading assayed the lowest in TiO_2 and as the specific gravity increased the trend was for the TiO_2 assay to increase, see Fig. 1. Since these results warranted further investigation, a 500-g capacity Torsion balance and 250 ml Le Chatelier specific gravity bottles were obtained.

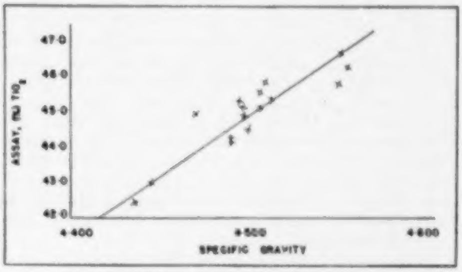

Fig. 1—Initial results, specific gravity assaying.

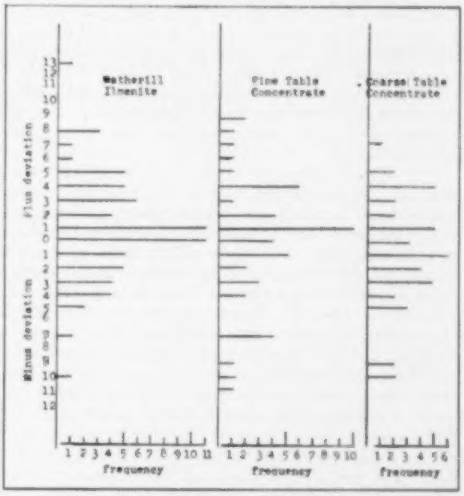

Fig. 2—Deviation vs frequency charts.

Shift samples of fine table concentrate, coarse table concentrate, and final ilmenite were tested. Each sample was split and 85 g weighed on the Torsion balance. The Le Chatelier bottle was filled with water to a zero mark. To avoid wetting the neck of the bottle it was found necessary to do this

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Table I. Specific Gravities and Average Proportions of Minerals in MacIntyre Ore		
Mineral	Avg Wt, Pct	Specific Gravity
Magnetite	35.0	5.2
Ilmenite	32.0	4.5-5.0
Plagioclase	10.0	2.6-2.8
Garnet	8.0	3.8
Pyroxene and Hornblend	8.0	3.0-3.6
Biotite	4.0	2.5-3.0
Spinel	2.0	3.6
Pyrite	0.5	5.0
Calcite	0.3	3.0
Apatite	0.2	3.2

Table II. Specific Gravity Standards

Wetherill Concentrate		Fine Table Concentrate		Coarse Table Concentrate	
Ml	TiO_2, Pct	Ml	TiO_2, Pct	Ml	TiO_2, Pct
18.42	46.5	18.74	42.5	18.72	43.0
.52	46.0	.84	42.0	.82	42.5
.62	45.5	.94	41.5	.92	42.0
.72	45.0	19.04	41.0	19.02	41.5
.82	44.5	.14	40.5	.12	41.0
.92	44.0	.24	40.0	.22	40.5
19.02	43.5	.34	39.5	.32	40.0
.12	43.0	.44	39.0	.42	39.5
.22	42.5	.54	38.5	.52	39.0

by means of a funnel with a long stem. After the water was added a zero reading was taken. A short-stemmed funnel was then used to add the weighed sample. To remove dissolved air from the water and thus prevent a possibility of erroneous figures, all water used in the determinations was boiled and cooled prior to use.

When the sample was poured into the bottle some air was carried with it. It was necessary to shake all the air from the grains before the volume reading was taken. When the air was removed the water displacement was noted and the flask emptied into a pan. The sample was dried and sent to the laboratory for chemical analysis.

Several hundred determinations were completed on the three products under investigation. Results showed that each product must have a separate standard.

The specific gravity tables, see Table II, were drawn up in the following manner. The volume obtained from a sample assaying high in percent TiO_2 was subtracted from the volume obtained from a sample assaying low in percent TiO_2. This value, that is, difference in volume, was divided by the difference in pct TiO_2 to obtain a scale showing the correlation between the specific gravity and the chemical assay. Since the same weight of 85 g is used in all tests, difference in volume is directly proportional to difference in specific gravity. Volume differences were used to facilitate reading standard charts.

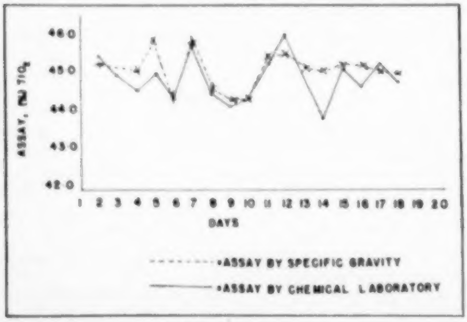

Fig. 3—Chemical vs specific gravity assaying.

The data shown below were used to determine the scale for Wetherill ilmenite, where each 0.1 pct difference in assay equals 0.02 ml volume difference.

TiO_2, Pct	Volume Per 85 G
45.7	18.85
44.3	18.58
1.4	0.27

$0.27/1.4 = 0.193$ ml per 1 pct TiO_2
$0.27/1.4 = 0.0193$ ml per 0.1 pct TiO_2
$0.27/1.4 = 0.02$ ml per 0.1 pct TiO_2

The 85-g samples used for the specific gravity tests were sent in for chemical assay. This chemical assay was presumed to be correct and the value obtained from specific gravity tables was recorded as either plus or minus the difference of the two values. If a value of 45.2 pct TiO_2 was obtained by specific gravity and the chemical assay was 44.8 pct TiO_2, the deviation was recorded as +4 units, each unit equaling 0.1 pct TiO_2. A check sample by specific gravity showing 44.7 pct TiO_2 would have a deviation of −1 unit.

The scale of 0.02 ml per 0.1 pct TiO_2 was used to draw up the standard table for Wetherill ilmenite. The plus and minus deviations were then found and the scale was shifted until the total plus and total minus deviations were as nearly equal as could be obtained. A summation of these data follows:

Product	Sum of Plus Deviations	Sum of Minus Deviations	Determinations, No.
Wetherill ilmenite	135	80	70
Fine table conc.	77	77	44
Coarse table conc.	32	88	43

Graphs of deviation vs. frequency were used to show the accuracy of determining pct TiO_2 by determining the specific gravity. Deviation-frequency charts for Wetherill ilmenite, fine table concentrate, and coarse table concentrate are shown in Fig. 2.

Conclusions

This paper is presented to show what has been accomplished by the specific gravity assay method at one mill, and to acquaint the mining industry with the possibility of a new tool for operational controls. It cannot replace chemical analysis but is a quick and convenient method for use of mill operators as a product control. It must be remembered that this method is used as a guide, and it forecasts the trends accurately enough to warrant its use. The results of a month's specific gravity analysis vs chemical analysis of actual plant control is shown in Fig. 3.

Each product to be analyzed by the specific gravity method must be individually tested and a separate standard table made.

The specific gravity method, as related to MacIntyre ore, is not infallible. Several times the results became too erratic for proper control.

Acknowledgments

The writer wishes to thank P. W. Allen, Plant Manager, J. J. Strohl, Mill Superintendent, W. P. Jenkins, Mill Foreman, and J. Bassarear, Junior Metallurgist, for their help and advice in evolving the specific gravity technique for TiO_2 analysis.

Reference

[1] Frank R. Milliken: Metallurgy at National Lead Company, MacIntyre Development. *Trans.* AIME (1949) **183**, p. 101.

Gravity Investigations In The Iron River-Crystal Falls Mining District of Michigan

Fig. 1—Area covered in gravity investigations of Iron River-Crystal Falls mining district. Round Lake area in solid black.

by L. O. Bacon and D. O. Wyble

THERE has been considerable speculation among mining geologists and mining men in general as to the relative merits of gravity methods in iron-ore exploration. Most of the investigations which have been carried out in recent years have been done by mining companies, and few if any of the results have been published.

Three gravity surveys have been made in the Iron River-Crystal Falls area,[1,2] one by the U. S. Bureau of Mines, with inconclusive results, and the other two by Radar Exploration Co. for the Cleveland-Cliffs and the M. A. Hanna companies, with results not known to the writers. Both the last-mentioned surveys were detailed investigations, the stations being spaced as close as 10 ft along traverses,[1] and were made to determine the feasibility of locating iron ore by gravitational methods.

Because of the presence of a large body of ore already extensively drilled, Dr. Harold L. James of the U. S. Geological Survey, Iron River, Mich., suggested that a further test of gravitational methods of exploration be made in the Round Lake area of Iron County, see Figs. 1 and 2. This survey indicated a considerable regional gradient and led to a study of the surrounding gravity field and its relation to the major structural features of the syncline within which the iron formation lies.

The southwestern part of Iron County presents a series of hills and hollows elongated in a roughly north-south direction. The bedrock topography is of pre-glacial origin. Relief is moderate; a maximum elevation difference of approximately 350 ft was observed during the survey. Major drainage is dendritic with the larger streams: the Brule to the south, the Paint to the north, and the Iron in the central part. Hills are thickly grown with hardwoods, and lowland areas, not under cultivation, are usually covered by second-growth softwoods.

The area covered in this paper includes the southern half of Iron County, Mich., with a deep synclinal basin occupying the central part of the area, and the southeast corner of Gogebic County, immediately to the west of Iron County.

The upper Huronian iron formation, slates, and graywackes which form the synclinal basin rest upon a greenstone basement. The basin has not been completely outlined by exposures, drill records or other means, as outcrops are scarce and the thick mantle of glacial drift makes geological investigations extremely difficult.

Glacial drift varies in depth up to 300 ft or more. The drift is usually thickest under the hills and is thinnest in the depressions and along drainage courses where most of the bedrock exposures occur. The area over which gravity investigations were made is almost entirely covered by glacial material.

Bedrock of the basin consists of a complexly folded series of iron formation, slates, and graywackes. One of the series of slates has a high magnetic susceptibility due to the fine-grained magnetite it contains. This formation can be detected readily by magnetic methods and provides a marker horizon which can be easily followed. The magnetic slates are stratigraphically above the iron formation, and much of the geological interpretation of the area is based upon the magnetic anomalies they produce.

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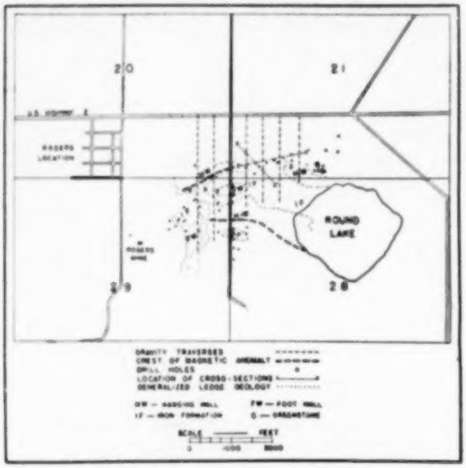

Fig. 2—Plan map of Round Lake area.

The iron formation, together with the slates and graywackes which occur in the basin, are considered to be Upper Huronian. This is younger than the major iron formation of the Marquette, Gogebic, and old Menominee Ranges. The age of the greenstone is in doubt. It has been suggested by Pettijohn[5,6] that it might be equivalent to the volcanics of the Middle-Huronian Hemlock greenstone or the greenstone of the Keewatin series. The greenstone, where exposed, has been observed to be generally massive and commonly characterized by pillow and agglomerate structures. In some places the greenstone is strongly magnetic and in others it contains interbedded slate with some magnetic iron formation.

The term *iron formation* includes the ore, hematite and limonite, cherty iron ore, jasper, sideritic slates, and minor ferrignous slates.

Because of the extremely complicated cross folds, overturned folds, faulting, and shearing, it is almost impossible to follow the stratigraphical sequence in a single section. Major folds are complicated by minor ones; even folding within folds occurs, and the structural footwall is not always the stratigraphical one. Geological interpretations based upon underground mapping and drill data are often in error because of the extremely complex structure. The stratigraphical succession as outlined to date by members of the U. S. Geological Survey[3-4] follows.

The lower part of the footwall series of rocks between the greenstone and iron formations consists mostly of graywacke, above which lies a dark gray sericitic slate. Both rocks contain 6 to 12 pct iron, occurring primarily as siderite. The thickness of these two members has not been determined. Overlying the sericitic slate is a highly pyritic, graphitic slate. This slate is the footwall for the iron formation throughout the Iron River district. Near the base of the graphitic slate is a breccia, the so-called *speckled gray*, which forms a marker horizon. The observed iron content of the slate varies from 2 to 25 pct. The thickness of the pyritic, graphitic slate is given as 20 to 40 ft.

The iron formation rests conformably on the graphitic slate. Where the iron formation is unoxidized it consists of dark gray chert and gray siderite with occasional layers of sideritic slates. Oxidation has changed the siderite to hematite in places, yielding an oxidized iron formation. The average iron content of the unoxidized iron formation is about 24 pct, whereas when oxidized the iron content is nearly 35 pct and the ore generally contains 50 to 60 pct iron. In general most of the iron formation is oxidized. Since the iron formation is tightly folded, its true thickness is hard to determine. Estimates run from 100 to 300 ft, but it has been found to be as thin as 10 ft in some places.

The hanging wall series, composed of graywacke, gray slate, and magnetic slate, lies disconformably upon the iron formation in some areas. Because of this disconformity the lower part of the hanging

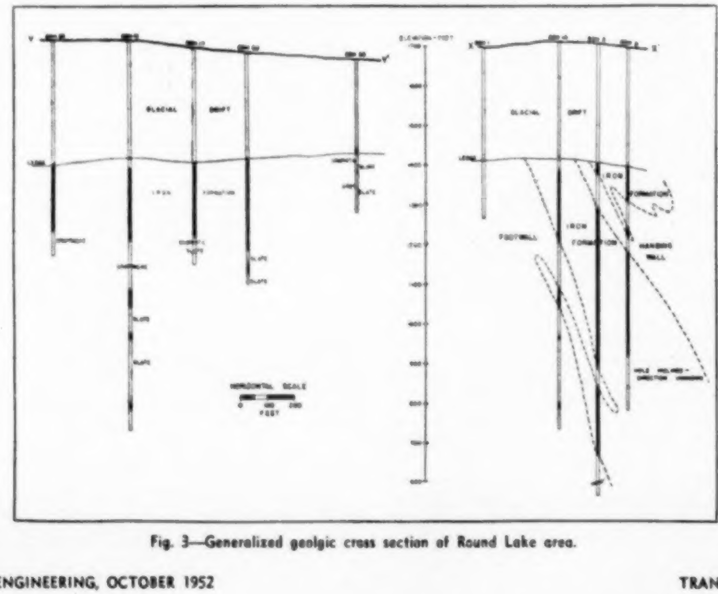

Fig. 3—Generalized geolgic cross section of Round Lake area.

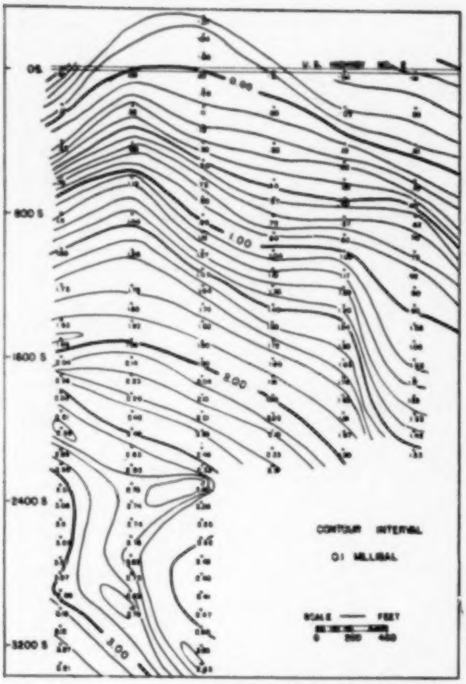

Fig. 4—Bouguer gravity map of Round Lake area.

wall series varies considerably in thickness and lithology. The contact between hanging wall and iron formation may vary from an extremely sharp demarcation line to a very slow gradational change. The graywacke and gray slate hanging wall members have a thickness of 100 to 400 ft.

Above the graywacke and gray slate members is a magnetic slate, the youngest rock in the area for which a description can be given. The lower portion of the magnetic slate is a gray, thinly-banded sideritic rock of cherty appearance quite often embodying finely disseminated crystalline magnetite, although some parts contain little or none. Above this occurs a dark, strongly magnetic rock composed mostly of a chlorite-like mineral, with a high percentage of minute crystals of magnetite scattered throughout. The iron content of this series runs from 15 to 25 pct and its thickness is estimated to be about 100 ft.

As previously mentioned, the ore deposits are mainly of soft red hematite and limonite with some hard hematite, the hematite being the more abundant. The orebodies are very irregular in shape and size, most of them occurring along the keels of synclines and apparently bearing a close relationship to the size of the associated folds. Some of the orebodies have been found immediately below the glacial drift and some have been drilled to a depth of nearly 2000 ft without being bottomed. The change from ore to iron formation of a percentage too low to be commercial is a gradual one.

The physical property of a rock of greatest importance in gravitational methods of exploration is density. Variation in the density of rocks causes a variation in the gravitational attraction which can be measured by a sufficiently sensitive instrument.

Densities of the rocks in the Iron River district according to Zinner[1] are given in Table I with the exception of the density of the greenstone, determined by Wyble[2] from 11 samples provided by the U. S. Geological Survey.

Table I. Densities of Rocks in the Iron River-Crystal Falls Area

Rock Samples	Density	No. of Samples from Density Profile	Average Density
Glacial drift			2.0
Magnetic slate	2.98		2.98
Graywacke	2.57-2.89	2	2.73
Iron ore	3.78,3.88,3.94	3	3.87
Cherty iron ore (jasper)	3.00,3.15,3.16	3	3.10
Black slate (footwall)	2.64-2.88	2	2.76
Graphitic slate	2.06-2.55	2	2.30
Greenstone	2.84-2.91	11	2.86

Density of the graphitic slate may be too low, as samples were taken from the dump where decomposition of the pyrite could have reduced the density materially. Because of the high pyrite content the rock in place has been estimated[1] to have a density of nearly 3.0.

Density of the iron ore is given from Table I as about 3.9, equivalent to 9.3 cu ft of ore per long ton. The mining companies, however, have found that 12 cu ft of ore per ton is more nearly correct.[1] If the density of the ore is computed on the basis of the above, the figure of 3.0 is arrived at. Since the composite density of the jasper, graywacke, and slates is also nearly 3.0, significant gravity anomalies over an orebody could not be expected. The cubic feet per ton factor is affected by a number of things besides the actual density. In arriving at this figure the following factors are also considered[2]: 1—Mining loss. 2—Rock occurrences in the orebody and the question of whether or not these are to be discarded in

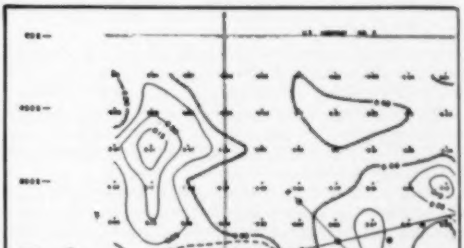

Fig. 5—Residual gravity map of Round Lake area.

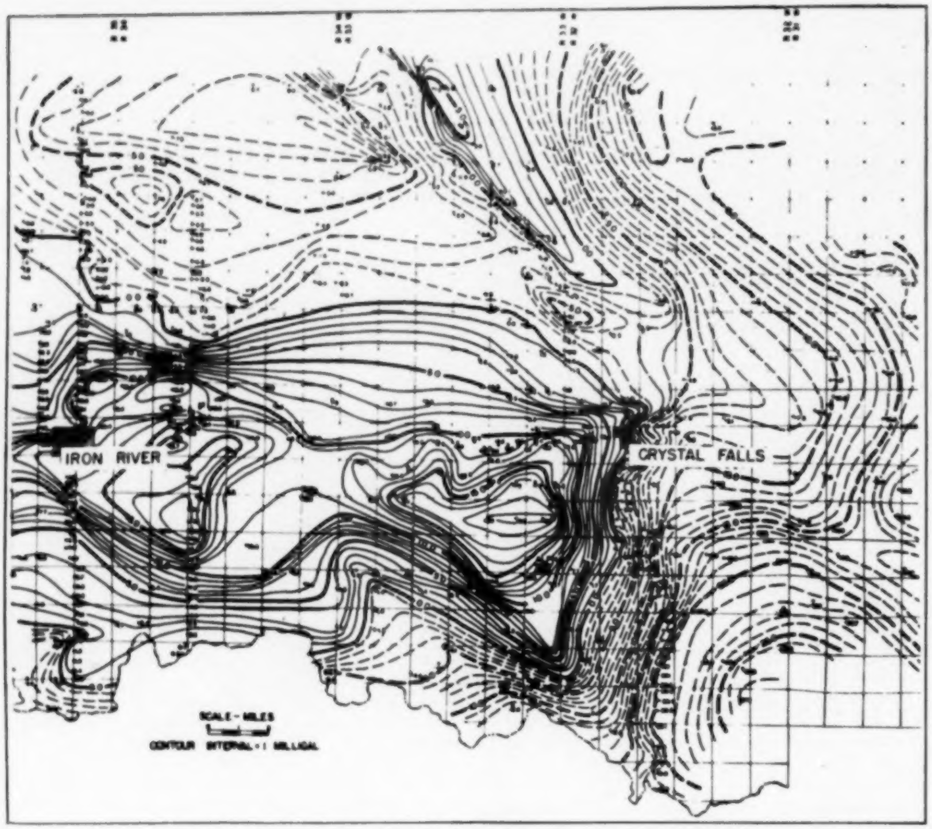

Fig. 6A—Bouguer gravity map of area between Iron River and Crystal Falls, Mich.

mining or go with the ore to increase the tonnage and adversely affect the grade. 3—Dilution with wall rock and/or material from the gob. 4—Occasional recovery of relatively clean ore from the gob resulting from mining loss in previous operations. 5—Gain or loss in moisture as a result of mining and subsequent handling.

Since a cubic factor of 12 takes into account the above items, it is obviously too high to use to calculate the ore density which probably lies, therefore, somewhere between 3.0 and 4.0. In view of the lack of data a figure of 3.3 has been assumed for the density of the iron ore. This results in a density contrast of 0.3 between iron ore and hanging wall-footwall series and of 0.2 between iron ore and cherty iron formation. The density of the latter is based upon only three samples. Royce[3] states that the density contrast of 0.3 seems reasonable as the ore encountered in their drilling was quite dense with apparently little porosity. The writers have assumed that there is a density contrast of 0.3.

The calculated gravity anomaly for Fig. 3 is 0.30 milligal, as determined by use of polar charts with end corrections applied for an orebody 1000 ft in length and for density contrast of 0.3. Residual anomalies smaller in magnitude are observed.

Field work was carried out with Worden Gravimeter No. 28, having a scale constant of 0.03483 milligals per scale division. The instrument has a fairly uniform drift which amounted to from one to two divisions per hr. Just prior to completion of the Round Lake survey, the instrument sustained damage in the field. After repairs were made the maximum drift had increased by a factor of three. The drift gradually decreased with time until it amounted to from 2 to 3 divisions per hr, or approximately 0.1 milligal per hr, near the end of the field season.

In the Round Lake survey, stations were set at 100-ft intervals along north-south traverses 400 ft apart, except along the northern edge of the area where the stations were 200 ft apart, see Fig. 2. Elevations were obtained with an accuracy of 0.1 ft relative to the gravity base station for which an elevation of 100 ft was assumed. Extreme care was taken to maintain the highest precision in the reduced data. The probable error of a single station is about ± 1 division, or ± 0.03 milligal, except in the southern portion where there is moderate relief. No terrain corrections were made on this survey.

In the regional survey, seven north-south traverses were laid out in the Iron River-Crystal Falls area with stations ¼ to 1 mile apart. Elevations for these traverses, to the nearest foot, were available from Michigan Dept. of Highway road profiles, or were obtained by leveling. Stations were also established at all points, usually road intersections,

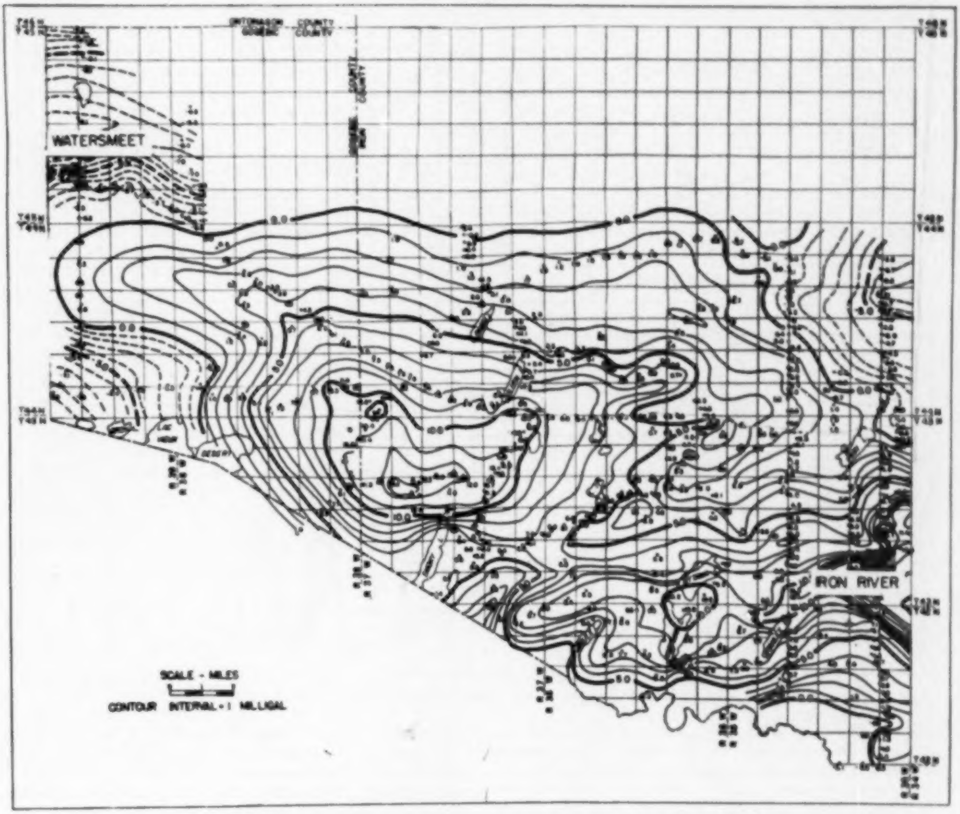

Fig. 6B—Bouguer gravity map of area between Iron River and Watersmeet, Mich.

which had an elevation given on the U. S. Geological Survey quadrangle maps.

In the Iron River-Watersmeet area about 150 station elevations were obtained by using an altimeter in conjunction with a station recording microbarograph to monitor barometric fluctuations. Elevation checks on U. S. Coast and Geodetic Survey bench marks up to 10 miles from the recording station indicated that 75 pct of the elevations were within $\pm$ 3 ft and that 95 pct were within $\pm$ 5 ft. A 5-ft error would cause about a 0.3 milligal error in the reduced data, a not too serious matter in a regional survey of this type. In all about 750 stations were occupied in an area of approximately 600 sq miles. This indicates a density of about 1 station per sq mile; however, this varies considerably, as much of the area is not readily accessible and has very low station density.

In reduction of the data, corrections were made for instrument drift, free-air effect, simple Bouguer effect, and latitude variation. No correction was made for terrain effects, since in the Round Lake area the terrain was relatively flat, while for the regional survey it was not necessary to make such a correction to maintain a precision of $\pm$0.3 milligal in the reduced data.

All gravity stations in the Round Lake area were referred to a local base station which was later tied into the regional survey. The regional survey was referred to a base station established at the U. S. Coast and Geodetic Survey BM Iron River, elevation 1515 ft, in front of the Iron River High School, and assigned an arbitrary value of 10.0 milligals.

Interpretation of Results of Gravity Investigation

The Round Lake Area: After reduction of the gravity data the results were plotted on a map, Fig. 4. This shows the Bouguer gravity uncorrected for regional effects. There is a strong regional gradient which becomes increasingly positive to the south. This regional effect obscures any gravity anomalies which may be the result of ore. The anomalies which apparently are associated with the ore or iron formation are revealed by changes in the contour lines. There is an indication of a positive anomaly between 0 East and 800 West traverse at 2000 to 3200 South. Another indication occurs at 1200 E near 1400 S. These indications are slight, however, and more or less obscured by the effect of the regional gradients. It is thus desirable to remove as far as practicable the regional effect from the Bouguer gravity.

The regional effect was removed by forming the residual gravity value by the nine point system, where the residual is equal to the value of the station minus the average value of the station and the eight closest stations on a uniform grid, in this in-

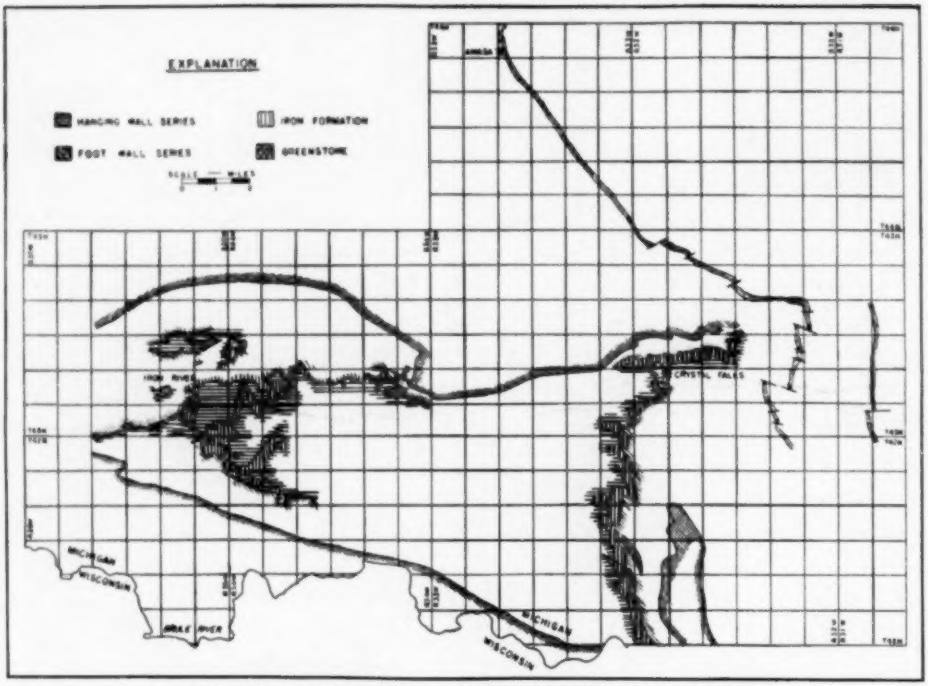

Fig. 7—Generalized geolgic map of Iron River-Crystal Falls District, Mich. After Dutton.[9]

stance a 200-ft grid system obtained by interpolation between traverses 400 ft apart. The residual gravity map is shown as Fig. 5 with pertinent geological data superimposed.

Anomalies are now apparent in the southwest, northeast, and northwest portions of the area. This agrees fairly well with the geological features as revealed by available drill hole information, there being in general a positive residual over the areas underlain by iron formation. The only area in which ore is known to exist is along section Y-Y'. This shows a somewhat greater residual value than where no ore was encountered. The positive residuals at 600 S, 400 W, 1800 S, 800 E, and 2300 S, 0 E have not been drilled, or if so the records are not available to the writers.

Without more work it is impossible to say whether the residual gravity shown in Fig. 5 is caused by the density contrast between iron formation and ore or by the contrast between iron formation and the foot and hanging wall because of the lack of density data.

Regional Gravity Investigations: Results of the regional gravity surveys are shown as two contour maps, Figs. 6A and 6B, with a contour interval of 1A and B milligal. This interval is too large and the station spacing too great to show any small local anomalies, but it does reflect the major structural features.

The gravity anomalies encountered are extremely large. In the southeastern portion of the area, Fig. 6B, there is a change of 46 milligals in a distance of about 10 miles. The negative anomaly lies between two outcrops of Laurention granite,[10] not shown on Fig. 7, and probably is caused by the lower density of the granite. No density values are available on this rock. The outcrop of greenstone in the southwest corner of T. 42 N., R. 32. W. lies in the region southwest of Alpha where the gravity gradient is the greatest. Its effect on the gravity values apparently merges with the effect of the granite.

The highest positive gravity anomalies apparently occur where the structural basin is the deepest. Considering the magnitude of the gravity anomaly between the area southwest of Crystal Falls and the area about 3 miles southwest of Amasa, which is underlain by greenstone, we can obtain an estimate of the thickness of the Huronion sediments. Taking the average values of the anomaly to be 25 milligals, and for simplification applying the formula for the gravity effect caused by a body large in horizontal extent and of finite thickness, then the difference in thickness is given by

$$\Delta h = \Delta g / 2 \pi \gamma (\Delta \sigma)$$

where g = gravitational anomaly, γ = gravitational constant, $\Delta\sigma$ = density contrast, Δh = thickness of anomaly producing source. For a density contrast of 0.3 between Huronion sediments and greenstone and for the 25 milligal anomaly, a value of 6500 ft is obtained as the necessary amount of rock of a higher density to produce that large an anomaly. This is probably not an unreasonable figure.

Another striking feature of this map is the large positive anomaly in the south central portion, coinciding very well with the known syncline containing the iron formation, Fig. 7.

The gravity map, Fig. 6A, indicates that the syncline narrows with a southeast trend near the Michigan-Wisconsin border, south of Crystal Falls. At

Crystal Falls there is a narrowing of the anomaly and several miles east of Crystal Falls the gravity trend makes about a 120° turn northwestward toward Amasa. Comparison of the Generalized Geologic Map, Fig. 7, with the Bouguer Gravity map of the area shows a very excellent correlation.

The gravity map shows a large re-entrant along the southern limb of the major syncline, 5 miles west of Alpha. This may be the result of a major fold, a projection of the greenstone toward the central portion of the structural basin, or possibly a major fault.

One of the small local anomalies of interest is the one occurring at the northern half of Section 20, T 43 N., R 34 W. This is about one mile northwest of the Round Lake area. To the writers' knowledge, no drilling has been done within the area outlined by the closed 12.0 contour lines. This area is almost directly east of the Bates mine, located in the NW¼ of the NW¼ Section 19, T 43 N., R. 34 W., and could well be underlain by an eastward extension of the iron formation which occurs in the vicinity of the Bates mine. Folding or faulting may have interrupted the continuity of the formation.

Another explanation of the local anomaly mentioned above is possible. Since the anomaly has a magnitude of only 2 milligals it could conceivably be caused by a change in elevation of the bedrock topography. A difference of approximately 160 ft in the bedrock topography could cause a 2-milligal anomaly as the density contrast between bedrock and drift material is about 1.0. Even though variations in the bedrock topography could cause this anomaly, the writers believe that because of the spatial relationship further investigation in this area is warranted.

Fig. 7 shows iron formation to the north of Iron River apparently not reflected on the gravity map. The writers have no explanation for this other than the fact that the depth of the synclinal basin here may be quite shallow and the paucity of data does not allow for delineation of any anomaly.

Of primary importance are anomalies occurring in the western portion of Iron County and the southeastern portion of Gogebic County, where little is known of the subsurface geology. The gravity anomaly narrows to the south and westward of Iron River but extends to Brule Lake a distance of about 10 miles. This anomaly coincides with the known trend of the iron formation as shown in Fig. 7, which in this part of the area is based on drill records. The anomaly may extend into Wisconsin from T 42 N, R 36 and 37 W, the closed contours being based on very few points. Of primary importance is the large gravity anomaly occurring in T 43 and 44 N, R 37 and 38 W. This anomaly has a magnitude about equal to that southeast of Iron River and covers a large area. In view of the correlation between the synclinal basin and the gravity anomaly between Iron River and Crystal Falls, this westerly anomaly may well be associated with another synclinal basin which may in turn contain iron formation and associated rocks.

There is no geological information available on this area although some magnetic work has been done north and west of Smoky Lake. The results of this magnetic work are not known to the writers.

Dr. Harold James states that there is no known iron formation in the area in which this gravity anomaly occurs. Since the glacial drift is thick and there are no outcrops known to the writers, this area may be underlain by a structural basin and may well be a potential new iron-ore district.

To sum up briefly, there is some possibility that gravity methods may be able to differentiate between large iron ore bodies and the iron formation in the Iron River-Crystal Falls district, although further investigation along this line is necessary to substantiate the conclusion.

The regional gravity work outlines quite clearly the major structural features of the Iron River-Crystal Falls synclinal basin.

A large anomaly occurring about fifteen miles west of Iron River may well be associated with another structurally similar basin and consequently may be another potential iron ore district.

Acknowledgments

The authors express their appreciation to Dr. Harold L. James for his assistance and comments on the gravity investigations as well as to Messrs. Kenneth L. Weir and Waldon P. Pratt, all of the U. S. Geological Survey, Iron River District, for field assistance in the Round Lake area.

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Periclase Refractories in Rotary Kilns

by Leslie W. Austin

ROTARY kiln operators will agree that some of the most severe conditions a refractory must stand occur in the hot zone of a kiln burning Portland cement, dead burn dolomite, magnesite, periclase, and similar materials. Frequently the operator is faced with factors beyond his control which drastically shorten the life of refractories. Shutdown due to mechanical failure can be serious if the period is of sufficiently long duration to cause the dropping of coating or the loosening of the lining. A change in slurry can affect the coating and cause ring buildup. A change in type of fuel and its effect upon the flame can cause a shift in location of the hottest zone. Weekend shutdowns or any other interruption can cause the operator trouble and may damage the refractories, since stopping and starting a rotary kiln is certainly more difficult than stopping and starting a motor. Some operators have tried to set an estimate of damage for each shutdown in equivalent days of running time.

Conditions affecting the refractory may be roughly grouped in four classes: chemical attack, mechanical stress, thermal shock, and abrasion.

Chemical Attack: The drive to obtain maximum production through a kiln demands maximum operating temperatures, temperatures which are limited more by the ringing up or melting of the clinker. This can cause interface temperatures at the junction of coating and refractory which require the use of a basic kiln block to withstand the chemical attack. Chemical changes take place within the refractory itself, especially in chemically bonded or unburned kiln blocks. These changes cause the formation of the ceramic or burned bond. Migrating liquids or fluxes from the kiln charge have an effect within the refractory and lead to mineral or glass formation. The alkalies, sodium and potassium, migrate into the refractory as silicates, chlorides, sulphates or other salts. They may move under capillary action or may be caused to move by volatilization with condensation in the cooler portion.

Mechanical Stress: Concentrated stress may be caused by several factors or combinations thereof. 1—The rings of refractories must be kept tight and rigid within the kiln, and this alone demands considerable force to hold the blocks in place. So that the force will not be concentrated, the blocks should fit the circle as perfectly as possible, with the faces in contact overall. 2—As the kiln is heated, thermal expansion takes place at the hot end of the kiln block. Since this disturbs the plane face it too can cause a concentrated stress at the two ends of the block, and shearing stress can be set up within the brick itself because of the difference in expansion between the two ends. 3—If a lining becomes loose and moves in the shell very severe stress can be set up, and as the kiln rotates this load changes and gives the effect of repeated loading. Permanent expansion of the refractory can also cause severe loading. 4—Not least important, flexing of the kiln is frequently the cause of concentrated stresses.

Thermal Shock: Thermal shock, the result of heating and cooling too rapidly, occurs on starting and stopping or when a large patch of coating drops, exposing the bricks. Again, its destructive effect is often the result of phase change, liquid to solid or the reverse; dense refractories loaded with glass-forming impurities are particularly susceptible. Thermal shock is a problem with refractories set in the wall or roof of a stationary furnace, and becomes even more serious in a rotary kiln, the tendency to spall being magnified with movement and concentration of stress. Uniform rate of feed and loading insures both better coating and a more uniform stress.

Abrasion: If the refractories do not take a coating, abrasion can become a most destructive factor. Movement of the lining in shell or movement of loose blocks causes abrasion, which is also most destructive if the refractories do not take a coating.

An analysis of the problem of basic lining for the hot zone reveals, therefore, a number of desirable characteristics: high refractoriness, basic chemical reaction, resistance to spalling, good strength at all stages, ability to take coating, true sizing, volume stability, and abrasion resistance. Increased demand

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Fig. 1—Relative size of volatized silica as shown by electron microscope. X30,000. Average particle size approximately millimicrons.

for cement production and the improvement in basic kiln blocks have led to the substitution of basic refractories for fireclay and 70 pct alumina refractories in the hot zone of rotary cement. Although each kiln presents an individual problem, even in the same plant, there are basic characteristics inherent in all. If the clinker and the refractory lining react together to form a liquid slag at the operating, or clinkering temperature, the operation is in jeopardy if the kiln is overheated or the coating dropped. Much time and production can also be lost if the operation is necessarily controlled by the temperature of reaction between the lining and the kiln burden instead of the temperature for optimum production rate. Basic kiln blocks have been known for some years as the best solution to the problem. Burned magnesite brick were the first basic lining material and in some places still are. However, because of their moderate ability to stand load at high temperature and their tendency to spall, magnesite brick never became widely used for this purpose. Large scale use of *magnesite-chrome* brick, by far the most important type of basic refractory for this service, began about 20 years ago. Initially the magnesite was the dead-burned natural product, but eventually pure synthetic grades formed chiefly from high purity magnesium hydrate and known in the trade as *periclase* came to be used. These furnish ideal grain for the formation of the forsteritic bond.

It is possible to make a periclase from precipitated magnesium hydrate which will be of relatively higher purity and more uniform composition. Deleterious impurities are eliminated, desirable additives are made, and the mass is burned at higher temperature than possible with the ordinary natural magnesite. One method of making periclase is to treat calcined dolomite with sea water. The magnesium hydrate is derived from the magnesia inherent in the dolomite and magnesium chloride from the sea water. The lime is removed as soluble calcium chloride. The filter cake of magnesium hydrate receives desirable additive and is burned to 1700°C or more with a resultant grain of low porosity and high refractoriness. This periclase is used as source of magnesia in both chemically bonded and burned blocks with or without the addition of chromite. Periclase of this type contains about 91.5 pct MgO.

Except for kilns making white cement, refractory grade chromite is used as a component of most basic kiln blocks in combination with the periclase. The chromite is a desirable component since it results in low thermal expansion, lower rate of heat conductivity, increased resistance to thermal spalling, and greater control of volume stability.

Taking into consideration inherent characteristics of periclase and chromite, and conditions under which refractory kiln blocks must operate, research was concentrated upon developing highly refractory crystalline bonds which would develop at relatively low temperatures in chemically bonded refractories and would give strength, volume stability, and resistance to spalling. The result of this work was a crystalline forsterite bond which forms in place by solid phase reaction between fine magnesia and what research workers have called *volatilized silica*. This silica, formed as a byproduct of ferro-silicon manufacture, may be made synthetically and is especially adaptable for use in refractories. The electron microscope, see Fig. 1, shows the particles to be nearly perfect spheres with an average diam of about 1/6

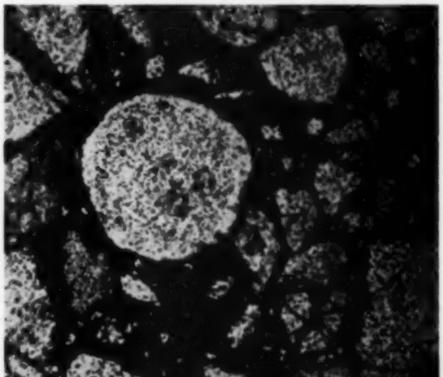

Fig. 2—The light-colored periclase grain is here clearly distinguishable from the dense bond.

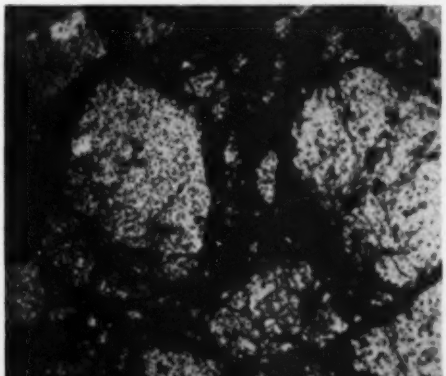

Fig. 3—After being fired to 1000°C the bond portion is lighter, but grain and bond are still sharply defined.

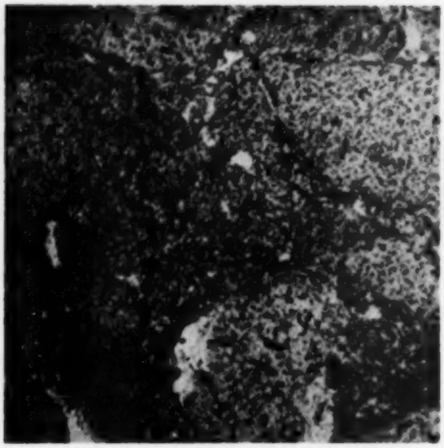

Fig. 4—After a burning temperature of 1500°C the periclase grains are surrounded by a well-developed crystalline forsteritic bond.

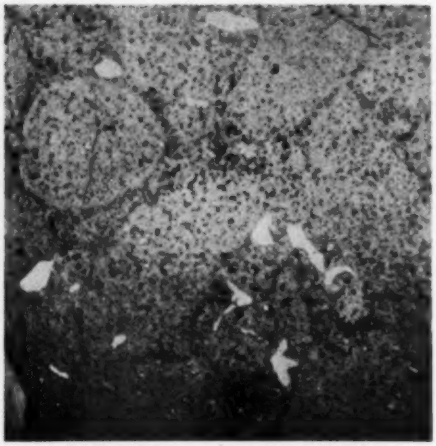

Fig. 5—Photomicrograph of a specimen of brick fired to 1800°C, illustrating high refractoriness of crystalline bond.

micron. The silica is mixed with fine magnesia in close approximation to the forsterite ratio, $2MgO \cdot SiO_2$, to form the bond.

Importance of the particle size of silica employed in the bond is shown in Table I, which correlates results of crushing tests on fired and unfired cubes cut from bricks similarly prepared except for the sizes of the silicas used. The notable improvements in bonding strength that accompany increase in the specific surface of silica used in the bond demonstrate the value of employing the finest silica economically feasible. As a basis for comparison, results are included in Table I for similar tests on a chemically bonded periclase brick using −200 mesh silica flour, note sample 6.

In Table I the sizing of silica is reported in terms of specific surface, as this presents a more readily understandable picture of the nature of materials so finely subdivided. For comparison, the specific surface of the finest Portland cement, an impalpable flour, is approximately 2800 sq cm per g.

Table I. Crushing Stress Developed by Test Specimen Made of Periclase Grain Bonded with Silica of Different Particle Size, Burned to Indicated Temperatures, Cooled, and Crushed. Values are Rounded Off to the Nearest 50 PSI

Sample	Specific Surface of SiO_2 in Bond, Sq Cm per Cm	Unfired	1000°C	1200°C	1450°C
1	67,500	17,500	6560	11,000	17,750
2	52,500	15,500	4850	8900	17,750
3	37,450	13,600	4100	7300	15,550
4	22,450	12,800	1400	4100	15,750
5	7410	11,200	1050	2890	13,600
6	(−200 mesh) under 2800	2600	1100	1400	3000

As stated above, the magnesia and silica are used in their combining ratio to form forsterite, that is, two mols of magnesia to one of silica, or for practical purposes, 134 parts magnesia to 100 parts silica. These ratios apply to the pure materials; when minor amounts of impurities are present it may be desirable to calculate from the analyses the additional amounts of magnesia required to convert these impurities to the higher temperature forms.

It is not necessary that the total magnesia or periclase in the brick bear a ratio to the silica which is in the forsterite ratio. Since there is no melting, the surface only of the periclase grains will be a source of magnesia to form the forsteritic bond. Magnesia over and above that required to form forsterite is not objectionable. Attention is called to the crushing strengths obtained by firing the specimens to intermediate temperature, cooling, and crushing. These tests show very clearly that even though the chemically bonded material loses some strength as it is heated, the specimens with the fine silica retain a much higher strength than those with coarser silica, when burned to intermediate temperature, and the strength increases rapidly to a high burned strength. This is of utmost importance in a chemically bonded kiln block.

Fig. 1 illustrates the relative size of the volatilized silica as shown by electron microscope. Figs. 2-5 are photomicrographs of thin sections made during the development of the forsteritic bond. These four slides were cut from a periclase brick bonded with fine silica and fine magnesia and burned to four different temperatures. The bond material contained a small amount of carbon and appears very dark in the chemically bonded or unfired specimen. Temporary bonds to give green strength for manufacturing and handling are obtained by the use of either magnesium sulphate or magnesium chloride.

Fig. 2 shows clearly the periclase grain as the light-colored portions. Each one of these periclase grains consists of many individual crystals. Attention is called to the close dense bond. The section shows a clear distinction between grain and bond. Fig. 3 is similar to Fig. 2 except that it was fired to 1000°C before the slide was made. The bond portion has lost much of the dark coloring, but the two-component system, grain and bond, is still clearly seen. Fig. 4 resembles the two preceding illustrations, with a difference in firing temperatures only. Here the burning temperature was 1500°C. It is much harder to distinguish between the grain and bond, but closer examination shows clearly the periclase grains which are now surrounded with well-developed crystalline forsteritic bond. Attachment of the bond to the crystals should be noted as well as

the remarkable volume stability of the bond. Examination of such slides under crossed nicols shows more clearly the distinction between grain and bond. A few small voids are shown, but they are not connected and are unimportant. The last slide in the series, see Fig. 5, was made from the specimen of brick fired to 1800°C or 3272°F. Even after firing to this high temperature the grains and bond are still clearly apparent and show that the bond is a highly refractory crystalline mineral bond which did not melt. Here again there are a few small unconnected voids.

These slides demonstrate the volume stability, bonding power, and high refractoriness of this bond which forms within the chemically bonded and burned kiln block without a volume change.

Combustible shims to compensate for thermal expansion are important, as are the steel spacers binding the kiln blocks together. Extensive research was carried out to develop the proper use of chromite in the basic kiln block. In this development work it is necessary to use the steel shim between the blocks in a manner approximating as nearly as possible that used in the kiln, since the steel plate oxidizes and has a swelling influence upon the chromite. Trials showed the importance of percentage chromite used, screen size of the chromite grains, and the importance of placing strong forsteritic bond as a protective coating around the chromite grains. The effect of temperature in influencing the volume stability of the periclase-chromite kiln block is also very important. Best results have been obtained by using the same starting formulas and processing for both the chemically bonded and burned kiln blocks.

In summation, performance records obtained over the past eight years have proved the advantages of using high purity periclase and the crystalline forsterite bond.

Application of Coarse Coal Magnetite Separators In An Existing Circuit

by V. D. Hanson, W. K. Heinlein, and J. M. Vonfeld

TWO overfeed drum-type separators using a suspension of magnetite in water as the separating medium have been installed in the Champion No. 1 preparation plant of the Pittsburgh Coal Co., Division of the Pittsburgh Consolidation Coal Company. A simplified flow diagram of the Champion No. 1 coarse coal circuit is shown in Fig. 1. A railroad car rotary dump, capable of handling 20 cars per hr, discharges the coal into a hopper. Coal is fed from the hopper of the rotary dump to a 60-in. belt conveyor by two reciprocating pan-type feeders. The belt conveyor feeds the run of mine coal onto the main raw coal screen. The first deck is fitted with 3 in. round hole perforated plate which removes the 3x0-in. coal. The second deck of the screen is equipped with 6-in. lip screens which separate the remaining coal into +6 in. and 6x3 in. The +6 in. coal is transferred to a shaking picking table and then discharged onto the loading boom. The 6x3-in. coal is fed to the No. 2 dense medium vessel for cleaning. The clean coal from this unit flows to a vibrating screen where the medium is removed and the coal sized into 6x4 in. and 4x3 in. The 6x4-in. coal is transferred to the loading booms and the 4x3-in. is conveyed to the clean coal classifying screens to remove degradation.

The 3x0-in. raw coal is blended in a system composed of three storage bins having a total capacity of 400 tons. From these bins the coal is fed onto a 48-in. flight conveyor by three reciprocating feeders. This flight conveyor delivers the coal to the Rheo plant. The 3x0-in. coal is then cleaned in two 48-in. Rheolaveur launders each equipped with two Rheo boxes. The first set of boxes removes a primary refuse and the second set produces a middling product which is recirculated in the launders. The cleaned coal and water overflows from the primary launders to two sizing and dewatering shaker screens where the water and − ⅜-in. coal are removed for further cleaning in the Rheo fine coal plant. The 3x⅜-in. clean coal is combined with the 4x3-in. coal from the No. 2 vessel, sized into 4x2 in., 2x1¼ in., and 1¼x⅜ in., then sent to the loading booms.

The 3x0-in. refuse from the primary launders is normally fed to the No. 1 dense medium circuit, but any portion or all of this material can, as an alternate, be processed in the Rheo rewash launder. The primary refuse when fed to the dense medium circuit is screened into ⅜x0 which is cleaned in the fine coal plant and 3x⅜ for cleaning in the No. 1 vessel. The clean coal from this vessel is passed over a vibrating drainage and rinse screen, then sluiced to the rewash sizing shaker where it is dewatered and sized for loading.

The installation of a pilot plant separating vessel using a suspension of magnetite in water was pro-

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Discussion on this paper, TP 3385F, may be sent (2 copies) to AIME before November 30, 1952. Manuscript, April 14, 1952. New York Meeting, February 1952.

posed for the Champion No. 1 plant early in 1941. Rising labor rates, coupled with the difficulty of hand picking 6x4-in. raw coal, in which the percentage of impurities was increasing, made the investigation of this process attractive. Plans for a pilot plant, however, were not completed until early in 1945. Later in the same year a cooperative arrangement was entered into with the Link-Belt Co. for the erection of the first magnetite dense medium plant to clean bituminous coal in this country. Construction was completed and the plant was placed in operation in January of 1946.[1]

The primary objectives were to discover whether the drum-type vessel could make a satisfactory separation, to determine the operating characteristics of the vessel and its auxiliary units, and to obtain information which would be useful in the design of new plants.

The original No. 1 dense medium vessel was designed to make a three-product separation. It was divided into two compartments by a partition and the flow of medium could be controlled so that upward currents could be created on one side and downward currents on the other. The vessel when operated in this manner could produce a clean coal with approximately 1 pct sink at 1.60 sp gr and a refuse containing 1 pct float at the same gravity, but the middling product was mediocre, since it frequently contained over 50 pct of 1.60 sink material. In an effort to solve this problem, fluid flow studies were made using a transparent model of the vessel, and although valuable flow information was obtained, a solution was not found. After this unit had been operated for approximately one year, the partition was removed and the vessel was employed as a two-product separator.

The experimental vessel which had been in service for 6 years was rebuilt in January 1952. A phantom view of the present No. 1 unit is shown in Fig. 2. It consists of a cylindrical steel casing 15 ft in diam, 13 ft wide, and 6 ft long. The wheel, or drum, which has given the vessel its type designation, has an outside diam of 12 ft and an inside diam of 9 ft. Sixteen flights divide the inside of the drum into compartments each having an approximate capacity of 6½ cu ft. The drum, supported inside the casing on trunnions, is driven by a 10-hp motor and speed reducer through a roller chain drive. The roller chain engages hardened steel teeth which are inserted in slots and spot-welded on each side of the drum. Wear on the chain is minimized by keeping it well above the medium level, so that the drum speed of 1 1/10 rpm allows ample time for the suspension to drain from the sprocket teeth.

The feed is sluiced down the feed chute by a stream of medium and enters the vessel. Separation takes place rapidly, the float coal being transported across the vessel and over the weir by the overflowing medium, most of which is admitted through the bottom of the vessel. The coal and medium travel first across a stationary wedgewire screen where part of the medium is removed. The remainder goes on to the first section of the vibrating screen, where it is drained off. On the balance of the screen area, the magnetite adhering to the coal is rinsed off and the product dewatered. The material which sinks drops into the compartments of the drum and is transported over a curved plate to the top of the vessel and discharged into the refuse flume. From the refuse flume the material drops to a drain and rinse screen. Fig. 3 is a view of the original installation at the operating floor level. The drives for the clean coal and refuse screens are in the left foreground. The top of the vessel is shown on the right and the densifier for magnetite concentration and storage can be seen in the background.

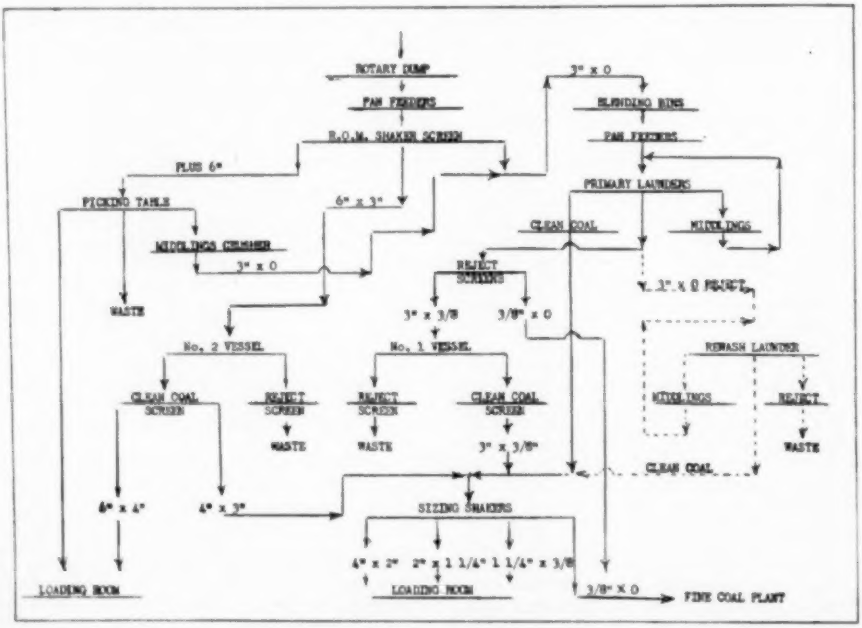

Fig. 1—Coarse coal flow diagram Champion No. 1.

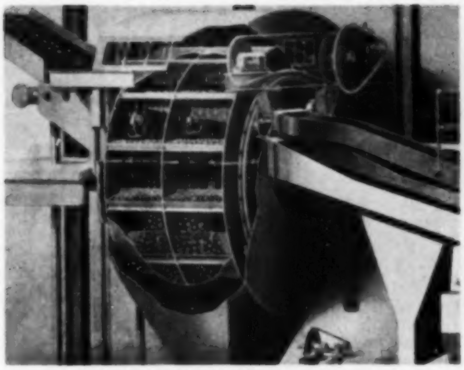

Fig. 2—Link-Belt float-sink concentrator vessel.

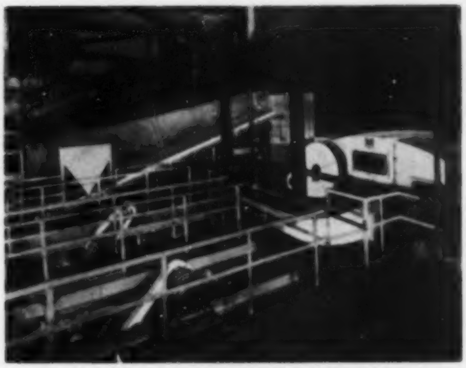

Fig. 3—Link-Belt heavy media coal washing equipment at the Pittsburgh Coal Co., Champion, Pa.

Fig. 4—A section of the +6-in. picking table.

Fig. 5—Coal discharging from the No. 2 vessel onto a vibrating screen.

After two years' experience with the No. 1 vessel, test information proved that a second vessel for treating 6x3-in. raw coal would make a valuable addition to the plant circuit. In the original layout, two tables were required, one for picking the + 6-in. material, and the other for the 6x4-in. size. Fig. 4 gives a close-up view of the present + 6-in. picking table. In Fig. 5 float coal is shown discharging from the No. 2 vessel onto a vibrating screen. The reject material is discharged into the flume directly above the clean coal overflow and then transferred to the refuse screen.

The flow of medium in the vessels and recovery circuit is shown in Fig. 6. Each vessel is supplied with medium from a pump sump. The No. 1 vessel requires 1300 gpm and the No. 2 vessel 1900 gpm. This medium is constantly circulating and its specific gravity must be kept two-hundredths below that of the desired gravity of separation.

All fresh magnetite is added to the system by a vibrating dry feeder to the No. 2 vessel sump. The magnetite used at Champion is purchased in 100-lb bags ready for use and ground so that 100 pct passes 65 mesh, 90 pct passes 150 mesh, and 60 pct 325 mesh. This magnetite as received contains approximately 86 pct of magnetic material. It is important to add magnetite to the pump sump so that some benefit can be obtained from the 14 pct of non-magnetic material.

For adequate rinsing of the coal, 2 gpm of water per ton of feed per hr are required. In the Champion No. 1 circuit, this means that the magnetite must be recovered from 900 gpm of spray water. Three magnetic separators operated in parallel series are used for recovery. The two separators in series are fed the underflow from the thickener containing 6 pct of solids and the third separator has a feed containing 3 pct of solids. These solids contain 32 pct of magnetic material. From the separators the recovered magnetite flows to two screw-type classifiers for thickening and storage.

Each vessel operator checks the density of the circulating medium at 15-min intervals with a specific gravity bottle. To increase the density, he can lower the screw in the densifier, thus adding more magnetite to the system. For lowering the gravity, water is introduced into the circulating pump sump. The operators of this system can easily maintain the density of the medium within one one-hundredth of the specific gravity desired.

Magnetite consumption for 1951 at Champion was 8 to 10 lb per ton of feed, or, since the combined recovery is 53 pct, the consumption per clean coal ton was 1.5 lb.

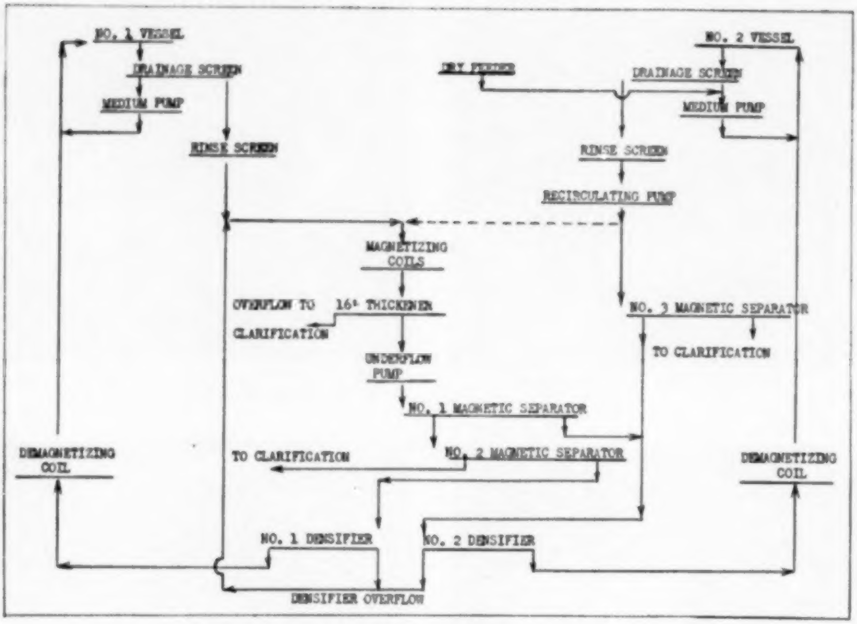

Fig. 6—Medium flow diagram.

Specific gravity analyses of the feed-washed coal and refuse from the No. 1 vessel when handling 300 tons of 3x⅜-in. primary refuse are shown in Table I. Calculated distribution of specific gravity fractions is shown at the bottom of the table. Efficiency for this test, calculated from the Fraser and Yancey formula,[2] was 95.8 pct. A distribution curve of the type developed by Yancey and Geer[3] is shown in Fig. 7. The curve crosses the 50 pct distribution line at 1.52 sp gr, which is, therefore, the gravity of separation.

This curve shows that none of the low gravity coal was lost in the refuse, but that some of the heavy gravity sink did report to the clean coal.

Table II shows specific gravity analyses for the No. 2 vessel when 175 tons of 6x3-in. feed are

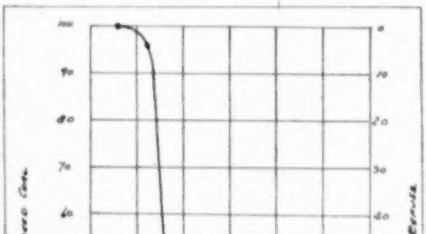

Fig. 7—Distribution curve for No. 1 vessel treating 3x⅜-in. primary refuse.

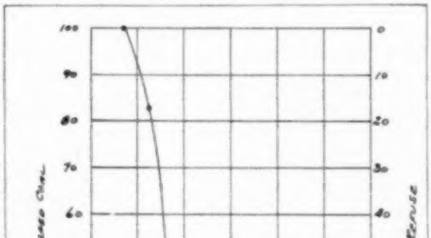

Fig. 8—Distribution curve for No. 2 vessel treating 6x3-in. raw coal.

Table I. Analyses of Feed, Washed Coal, and Refuse No. 1 Vessel Treating 3x⅜-In. Primary Refuse

Product	Sp Gr	Weight, Pct	Ash, Pct	Cumulative Weight	Cumulative Ash
Feed	Under 1.40	34.7	6.9	34.7	6.9
Calculated from	1.40 to 1.50	3.2	17.0	37.9	7.7
Products	1.50 to 1.60	2.2	22.2	40.1	8.6
	1.60 to 1.70	2.0	33.4	42.1	9.7
	1.70 to 1.80	1.8	38.3	43.9	10.7
	Over 1.80	56.1	77.9	100.0	48.5
Clean coal	Under 1.40	89.0	6.9	89.0	6.9
Weight 39 Pct	1.40 to 1.50	8.0	17.0	97.0	7.7
	1.50 to 1.60	1.3	24.0	98.3	7.9
	1.60 to 1.70	0.4	38.1	98.7	8.0
	1.70 to 1.80	0.2	47.0	98.9	8.1
	Over 1.80	1.1	74.2	100.0	8.9
Refuse	Under 1.40	0.0	0.0	0.0	0.0
Weight 61 Pct	1.40 to 1.50	.2	16.7	0.2	16.7
	1.50 to 1.60	2.8	21.6	3.0	21.3
	1.60 to 1.70	3.0	33.0	6.0	24.7
	1.70 to 1.80	2.8	37.9	8.8	28.1
	Over 1.80	91.2	77.9	100.0	74.4

Calculated Distribution of Specific Gravity Fractions

Sp Gr	Pct to Washed Coal	Pct to Refuse
Under 1.40	100.00	0.00
1.40 to 1.50	96.23	3.77
1.50 to 1.60	23.05	76.95
1.60 to 1.70	7.85	92.15
1.70 to 1.80	4.36	95.64
Over 1.80	0.77	99.23

Table II. Analyses of Feed, Washed Coal, and Refuse No. 2 Vessel Treating 6x3-In. Raw Coal

Product	Sp Gr	Weight, Pct	Ash, Pct	Cumulative Weight	Cumulative Ash
Feed	Under 1.40	60.8	5.3	60.8	5.3
Calculated from	1.40 to 1.50	4.1	16.7	64.9	6.0
Products	1.50 to 1.60	2.5	26.5	67.4	6.8
	1.60 to 1.70	.1	38.2	67.5	6.8
	1.70 to 1.80	.3	46.0	67.8	7.0
	Over 1.80	32.2	80.4	100.0	30.6
Clean coal	Under 1.40	93.5	5.3	93.5	5.3
Weight 65 Pct	1.40 to 1.50	5.2	16.4	98.7	5.9
	1.50 to 1.60	1.3	22.8	100.0	6.1
	1.60 to 1.70	0.0	0.0	100.0	6.1
	1.70 to 1.80	0.0	0.0	100.0	6.1
	Over 1.80	0.0	0.0	100.0	6.1
Refuse	Under 1.40	0.0	0.0	0.0	0.0
Weight 35 Pct	1.40 to 1.50	2.0	18.4	2.0	18.4
	1.50 to 1.60	4.6	28.4	6.6	25.4
	1.60 to 1.70	.4	38.2	7.0	26.1
	1.70 to 1.80	.9	46.0	7.9	28.4
	Over 1.80	92.1	80.4	100.0	76.3

Calculated Distribution of Specific Gravity Fractions

Sp Gr	Pct to Washed Coal	Pct to Refuse
Under 1.40	100.0	0.0
1.40 to 1.50	82.9	17.1
1.50 to 1.60	33.3	66.7
1.60 to 1.70	0.0	100.0
1.70 to 1.80	0.0	100.0
Over 1.80	0.0	100.0

treated. Calculated efficiency was 99.9 pct. The distribution curve, Fig. 8, crosses the 50 pct point at 1.52 sp gr. In this test, none of the coal lighter than 1.40 reported to the refuse and none of the material heavier than 1.60 entered the washed coal.

Three phase 60-cycle power is supplied at 440 v to the various motors in the plant. Direct current for the three magnetic separators and magnetizing coils is obtained from a motor generator set at 250 v. The total power consumed by the dense medium units is approximately 275 kw and is distributed as shown in Table III.

Accurate costs for the dense medium plant are difficult to determine since the men assigned to the

Table III. Power Distribution

Item	Pct of Total
Pumping	57.2
Screening	20.1
Conveying	12.8
Vessel Drives	6.1
Magnetic Separation	2.0
Thickening	1.8
Total	100.0

units perform some work chargeable to other accounts. Table IV includes only the direct cost of operating the vessels and their auxiliary equipment.

The effect of the installation of dense medium drums in the Champion No. 1 circuit is shown in the following series of charts.

Fig. 9 shows the pct weight of the output tonnage produced by the two vessels and the effect of progressive steps taken to obtain the maximum benefit from these units. In 1946, during the pilot plant stage, only 1.6 pct of the tonnage was dense medium clean coal. In 1947 the No. 1 vessel became part of the Rheo circuit, but owing to difficulties in disposing of the vessel refuse, only 40 pct of the primary launder refuse could be handled. A refuse belt conveyor was installed in 1948 and the percentage of total clean coal from the dense medium circuit increased 2½ pct. The effect of the installation of the No. 2 vessel in March of 1949 is shown by the increase in the percent of clean coal in 1949 and 1950. Finally, by the addition of more screening capacity for the 3x0-in. primary launder refuse, the No. 1 vessel was able to handle all of the 3x⅜-in. size of this product and a peak of 29.2 pct of the clean coal tonnage was produced by the dense medium system.

An improvement in the quality of all clean coal sizes has resulted from the installation, and Fig. 10

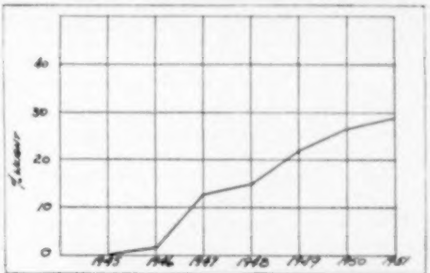

Fig. 9—Dense medium clean coal, percent of weight.

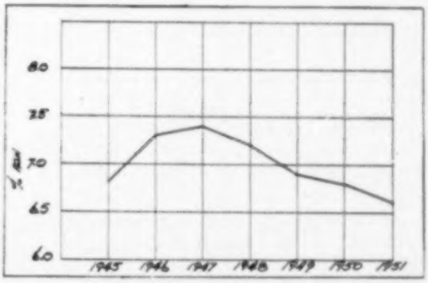

Fig. 10—Ash content of 4x2-in. clean coal.

Table IV. Cost of Operating Separators and Their Auxiliary Equipment

Item	$ Per Clean Ton
Operating Labor	.0488
Maintenance	.0323
Supplies	.0860
Power	.0117
Total	.1488*

* The above total does not include the royalty charge of $.01 per ton.

shows the change in ash content of the 4x2-in. size. This size was selected because it reflects the effect of both vessels. From 1946 to 1949, the No. 1 unit was recovering 4x⅜-in. coal from the refuse and from then on was handling the 3x⅜-in., while the No. 2 vessel produced 6x3-in. clean coal. The average ash content of the 4x2-in. for 1945 was 6.8 pct and rapidly increased with an increase in raw coal impurities to a maximum of 7.4 pct in 1947. In 1948 the trend was reversed and each successive year shows a gradual reduction toward the desired objective of 6.5 pct. The improvement in ash content is the combined result of the sharper separation in the dense medium vessels and better launder operation obtained by the reduction of the size and quantity of material fed to the primary Rheo units.

Fig. 11 shows the change in the feed and clean coal rates from 1945 to 1951. In 1945 the plant was handling raw coal at an average rate of 869 tons per hr and with a reject of 16.5 pct was able to produce 725 tons of clean coal. The chart clearly shows the increasing quantities of material that must be handled to maintain clean coal output.

In 1938 a production rate of 810 tons per hr was maintained with a feed rate of 888 tons; to attain the same rate in 1951, a feed of 1143 tons per hr was required.

The change in character of the raw feed is shown in Fig. 12. The reject from the raw coal of three of the mines supplying the bulk of Champion No. 1

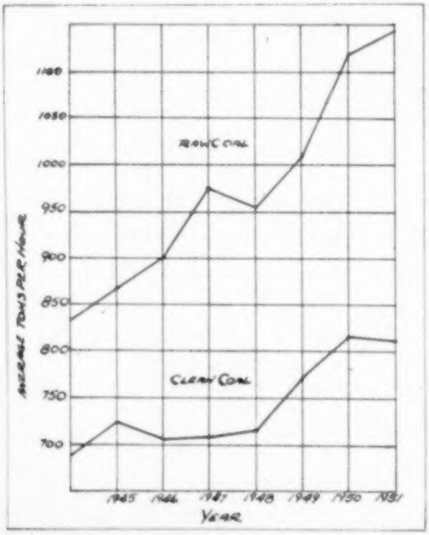

Fig. 11—Production curve for raw and clean coal, tons per hour.

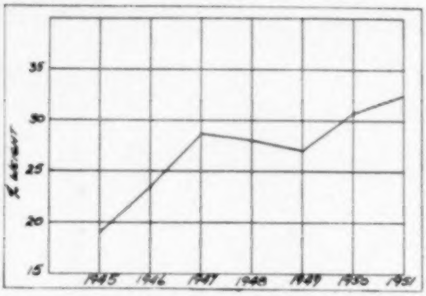

Fig. 12—Raw feed rejected from mechanically mined coal, percent of weight.

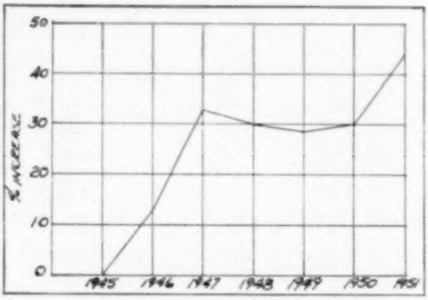

Fig. 13—Increase in clean coal produced per man-hour of mechanically mined coal.

tonnage has increased from 19.3 pct in 1945 to 32.3 pct in 1951. The increase in impurities in the raw feed is not, as might be supposed, the result of increased mechanical loading, but is instead the result of a change in the method of mining. The mines had found that it was economically more advantageous to load the slate with the coal rather than separately. The change in production rate resulting from this type of loading is shown in Fig. 13. From 1945 to 1951, there has been an increase of 46.3 pct in the clean coal produced per man hr. This increase in production has substantially reduced the labor cost per ton. It is believed that without the dense medium vessels this reduction could not have been made.

Acknowledgment

The authors are indebted to H. C. Rose, President of the Pittsburgh Coal Co., Division of Pittsburgh Consolidation Coal Co., for permission to present this information and wish to thank the members of the present staff and also those formerly associated with the Pittsburgh Coal Co. who have contributed to the successful application of the dense medium units. We wish particularly to thank J. B. Morrow, whose pioneering and foresight have contributed so much to coal preparation knowledge.

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aime NEWS

Declare Moratorium On New Accreditations

A 2-year moratorium on accreditation of curricula bearing new designations has been declared by Engineers' Council for Professional Development at its Executive Committee meeting on July 29, 1952. Reason: a more definitive description of what the contents of a curriculum should include to be appropriately designated as "engineering" is needed.

This action was brought on by the Committee of Evaluation of the American Society of Engineering Education which had become perturbed over the extension of the name "engineering" to numerous fringe curricula. The 2-year period will permit time for the Committee on Evaluation to study the question and make a recommendation for further action by ECPD.

ECPD action also provided that the following curricula should only be given provisional accreditation for one year upon either inspection or reinspection; such provisional accreditation to be extended annually for not more than two successive years pending the report: Engineering Physics, Engineering Mechanics, Geological Engineering, Geophysical Engineering, and Textile Engineering. There are 22 such curricula now accredited.

Accept Scholarship Offer From Socony-Vacuum Co.

An offer transmitted to the Institute by J. I. Laudermilk, of the Socony-Vacuum Oil Co., for the establishment of a scholarship in petroleum engineering for the academic year 1952 to 1953 was accepted at the May 21 meeting of the Executive and Finance Committees. The amount of the grant is $750. The Directors voted that it be administered by the Mineral Industry Education Div. of the AIME. This is the first scholarship to be given to the Institute for administration, in recent years at least. Several scholarships are, however, granted by the Woman's Auxiliary AIME, from their own funds.

The scholarship will be known as the AIME-Socony-Vacuum Scholarship in Petroleum Engineering. It will be granted to a student who expects to complete his academic work during the period of the Award, so is not open to those who expect to take graduate work.

The Hotel Statler, Los Angeles, will be headquarters for the 1953 Annual Meeting. The Hotel is said to be the most modern in the world. The artist's conception of the building gives some idea of its architecture and spaciousness.

Hotel Statler, Annual Meeting Headquarters For '53

The Statler Hotel, Los Angeles, newest addition to the Los Angeles skyline and possibly the most modern hostelry in the world, will be headquarters for the 1953 Annual Meeting of the AIME.

The hotel will provide the most extravagant surroundings to be found anywhere as the background for the outstanding technical gathering of the year. Every room is equipped with radio and television, and the hotel is completely air conditioned. Situated at Wilshire and 7th, the hotel is within the heart of the downtown area, and within easy traveling distance of theatres, golf courses, and the multitude of other attractions that have made Los Angeles one of the most fabled cities in the nation. The hotel also has a 435-car garage, which will make parking for those who drive to Los Angeles easy.

Plans for the technical sessions are rolling along, with the Mining Subdivision schedule almost completed. *The St. Joseph's Indian Creek Development* will be discussed by D. Kremer Bain, Harold Krueger, and Alvin A. Smith. *Block Caving to the Bottom of an Open Pit at Inspiration* will be presented by H. Carroll Weed and Charles Schwab. Other papers also have been scheduled.

The Iron Mining, Nonmetallic, and Joint Mining Geology sessions programs have been arranged. The Industrial Minerals div. program is also moving along on schedule. J. N. Grendell will present a paper on *Pressure Hydration of Lime*. The American Ceramic Society has agreed to present a producer-consumer paper on talc. Another paper on the agenda is *Flotation of Kings Mountain North Carolina, Spodumene Ore*. Mining Geology has announced that eight papers will be read at its sessions.

AIME Membership Card No Longer Dues Receipt

Decision has been made to send out the 1953 membership card to AIME members, in good standing, along with the regular year-end bill for dues. Heretofore no membership card has been sent until payment has been received. This necessitated a second mailing and sometimes resulted in delay at peak periods or when it was impossible to identify a payment without further correspondence, as often happens when payment is made by a bank or by an employing company. A saving to the Institute of possibly $1000 should result.

The membership card will not, therefore, constitute a receipt for dues as it has in the past. Canceled checks of members will constitute a receipt. When payment is made by means whereby the member cannot be sure that his dues have been received by AIME headquarters, a receipt will be mailed if requested. An evidence that AIME headquarters lists a member as unpaid is the non-receipt of journals beginning with the April issues. Members are cut off the mailing list in March if their dues have not been received.

It is hoped that this new practice will meet with the approval of members and will result in prompt payment of dues.

D. H. McLaughlin, 1950 president of the AIME, and Mr. and Mrs. Merrill Shoup, take time out from the round of activities at the Pikes Peak Subsection meeting the evening before the Saturday night banquet. Shoup is president of the Golden Cycle Corp.

B. Barret Griffith Calls For Return to Gold Standard at Pikes Peak Subsection Summer Meeting

B. Barret Griffith delivered the principal speech at the first summer meeting of the Pikes Peak Subsection, Colorado Section, of the AIME held at Cripple Creek. In his address, he advocated a more sound fiscal program for the U. S. and a return to the gold standard.

In connection with the speech, Dr. Donald H. McLaughlin, past chairman of the AIME, and president of the Homestake Mining Co., expressed confidence that the nation would soon return to the gold standard.

The chairman of the subsection, David A. Carter, welcomed the group and introduced the following to the meeting:

Merrill E. Shoup, president, Golden Cycle Corp.; M. I. Signer, dean of the Colorado School of Mines and a member of the board of the Colorado Section; Ben C. Essig, Colorado manager for Gardner-Denver Co., and a member of the Colorado Section Board; J. Clair Evans, president of the Denver Fire Clay Co.; Albert E. Seep, president of the Mine and Smelter Supply Co.; Harold S. Worcester, vice president of King Lease, Inc.; and Lute J. Parkinson, head of the Mining Dept., Colorado School of Mines.

AIME President Michael L. Haider (second from left) posed with officials of the Pennsylvania Anthracite Section preceding the annual summer meeting at Split Rock Lodge, on scenic Lake Harmony in the Poconos, on July 2. With him is Section Chairman D. C. Helms (left), Charles S. Kuebler (third from left) general chairman of meeting arrangements, and Floyd S. Sanders, perennial Secretary-treasurer of the Section. Over 250 members and their wives were present for the meeting which included a social hour, courtesy of the sponsoring coal companies and manufacturers, banquet, business meeting, and address by Mr. Haider. New Officers for the section are as follows: John M. Reid, Chairman; Edward G. Fox, Vice Chairman; and Floyd S. Sanders, Secretary-treasurer; and Franz Edgar Kudlich, Wilmot C. Jones, John W. Buch, Francis E. Sterner, and Ralph A. Lambert, Executive Committee.

Journals Schedule Special December Issues

John V. Beall, Manager of Publications, announced at the June 18th Board of Directors Meeting that both JOURNAL OF METALS and MINING ENGINEERING would publish special issues in December of this year. The JOURNAL OF METALS will again feature the annual Electric Furnace Steel Conference. MINING ENGINEERING will feature articles describing the various operations of the Chile Exploration Co., a subsidiary of the Anaconda Copper Mining Co., at Chuquicamata, Chile.

Mr. Beall also reported that while expenses had increased to the point where they might exceed the budget, increased advertising for the magazines would more than compensate for a possible deficit.

Joe B. Alford, Secretary of the Petroleum Branch, reported that the advertising income for the JOURNAL OF PETROLEUM TECHNOLOGY was increasing and would completely support the Journal in a few years.

Geology Subdivision Issues Program Call

The Geology Subdivision of the AIME is sending out a call to any member of the Institute who desires to take part in the MGGD program for the Annual Meeting to be held in Los Angeles next February.

Any member who wishes to present a paper is invited to contact E. L. Clark, Missouri Geological Survey, Rolla, Mo., chairman of the program committee of the Geology Subdivision and coordinator of the overall program of the Mining, Geology and Geophysics Subdivision.

Los Angeles Invites PAIMEG Meeting

The Pan American Institute of Mining Engineering and Geology (PAIMEG) is being specially invited to hold its fourth congress in Los Angeles at the time of the AIME Annual Meeting, Feb. 15 to 19, 1953. It is probable that PAIMEG's own program will immediately follow that of the AIME, with technical sessions on Friday, February 20. Members of PAIMEG will be allowed the same registration fee for the AIME meeting as is granted to AIME members.

Doherty Memorial Fund Increased By $50,000 From Benefactor's Estate; To Expand Activity Range

The Henry L. Doherty Memorial Fund, established in 1945, has received a gift of $50,000 from the estate of the late Mr. Doherty. For several years the Cities Service Co. has contributed $5000 annually in lieu of income from the Fund, but this is the first large gift to the principal of the Fund. A further $50,000 is expected shortly from three other sources to establish a total Fund of $100,000.

Mrs. Helen Lee Wessel, president of The Henry L. and Grace Doherty Charitable Foundation, Inc., made the gift in the form of Surface Combustion Corp. 5 pct debentures "as a contribution by this Foundation to American Institute of Mining and Metallurgical Engineers." Mrs. Wessel adds, "It is our further understanding you are contemplating that the income from the Fund will be used to finance the printing and distribution of publications, to grant scholarships, and for research in matters relating to the petroleum industry, all, to the extent feasible, within fields and along lines in which the late Henry L. Doherty was known to be deeply interested."

Heretofore, the annual contribution from the Cities Service Co. has been used for meeting the cost, in part, of the annual petroleum statistics volume and of the Buckley volume on *Petroleum Conservation*.

The committee administering the Fund has, since the Fund was established, consisted of John M. Lovejoy, Chairman, with Warren A. Sinsheimer and E. DeGolyer.

Fall Meeting Oct. 24

The Mining Branch of the Southern California Section of AIME is scheduled to hold its annual Fall Meeting at the Statler Hotel, Los Angeles, Oct. 24, 1952.

The technical session starts at 9:30 am. The program has been arranged to include:

Activation of Bleaching Clays, by R. S. Lamar, director of research, Sierra Talc and Clay Co.; *Geobotanical Prospecting for Uranium*, by Helen L. Cannon, U. S. Geological Survey; *Sulphurdale Sulphur Deposits*, by R. L. King, consulting engineer; *Flotation of Phlogopite*, by R. L. Cornell, vice president, California Testing Laboratory; *Johnson Camp Geology*, by Arthur Baker III, geologist, Coronado Copper & Zinc Co.; and a sound movie of atomic tests in Nevada, prepared by the U. S. Atomic Energy Commission.

THE DRIFT OF THINGS

by Edward H. Robie

A Company's Stake in the AIME

AT a recent meeting of the AIME Board there was considerable discussion of a suggestion that companies should be more interested in promoting AIME membership among their employes. The advocate of this idea argued that companies benefit greatly if their employes are active AIME members, so even on purely a selfish basis they should use various means to encourage membership. Also he argued that companies in general make no contribution to the support of the AIME although benefiting from its work, so should not object to making more of a contribution, financial or otherwise, than they do.

One Director stated that an investigation had developed the fact that a large majority of members felt it unwise for employers to pay the initiation fees or dues of employes. They felt that AIME membership should be on a purely personal basis; that the rewards of active membership were sufficient so that it is a good investment for a man in developing his own professional stature, presumably resulting in an increase in salary sooner, or in greater amount, than would otherwise be the case. If an employer paid a member's dues or initiation fee the member would take less interest in the Institute. But this Director felt that an employer should otherwise encourage his employes to join the Institute, attend its meetings, and write papers based on company work for presentation and publication. Another Director reported that his company pays the expenses of AIME members in its employ to meetings; the company also encourages nonmembers to attend AIME meetings but does not pay their expenses unless they present papers of interest to the company. Another Director, representing a supplier, stated that his company's policy was to send some of its staff to all AIME meetings at which papers of interest were to be presented or at which valuable contacts could be made.

The consensus seemed to be that an employer should preferably not pay the initiation fee or the dues of a member except in special cases. Such a special case might be that of a technical librarian whose company might thereby obtain technical publications at less expense than if purchased at the nonmember price. Another special case might be that of a company which wished to keep in touch with AIME activities in a certain field which might be only a fringe field for the employe for whom the membership was taken out. But companies should encourage their men to attend AIME meetings, and pay their expenses thereto. They should also help Local Sections in their territory to put on good meetings. They could do this by such means as encouraging the writing of papers; by contributing a part of the cost of dinners for Junior Members and Student Associates; by contributing transportation for a group in a company car; and by permitting and encouraging inspections of their plants on field trips, perhaps providing a luncheon for a group.

The Directors of the Institute have never looked with favor on company memberships in the AIME. But they do not frown on passing the hat for voluntary contributions for special and occasional purposes, such as contributing souvenirs to an Annual Meeting, or perhaps to financing a general cocktail party in preference to individual hospitality in hotel suites; or to contributions to worthy funds, such as for student prizes or scholarships, for awards or medals.

Members of the Institute who have risen to top rank in their companies can well give thought to ways in which they can further the operation of their professional society. Their employes, as members, will benefit, as will the companies themselves.

"The People" Are of Good Will

By the time this appears in print, the Olympic Games in Helsinki (pronounced with the accent on the first syllable the Finns tell us) will have been largely forgotten, but their success affords a lesson that should be remembered. It was not the many world's records that were broken. Rather it was a congress of people from all over the world that met in peaceful contest, with a spirit of good will and cordial fellowship. The hospitality of the Finns and their conduct of the meeting, with the meager resources of their little but much admired country, brought much favorable comment. But this atmosphere could not have been maintained had there been exhibitions of international and interracial enmity by the contestants. All got along beautifully with one minor exception and the fellowship between the two leading groups of contestants, the Russians and Americans, was most cordial.

The wrong people are running the world. If the athletes were put in power, or the Quakers, the ministers, the doctors, or the engineers, specially the mining engineers, we venture to say that wars would be less frequent and that people would lead happier lives. The politicians and the diplomats have a sorry record of ineptitude so far in the twentieth century. But do they ever apologize or admit their mistakes? No. They rather talk about increasing our facilities for what they euphemistically call our defense and security. Defense is too often merely another name for war; and there is no such thing as security in the measures commonly adopted to attain it.

Intelligence Test

Some of the young ladies who are compiling our Directory are being forced to learn something about geography, a subject, like grammar, so largely neglected in our schools. About a sixth of the AIME membership lives outside of the United States. Often it is a problem, never completely and successfully solved by those who work on our Directory, whether a country is independent or merely a state or province in another country. Geographical ignorance even enters the door of the Secretary's office. We read the other day that Viet-Nam is the eleventh country to ratify the UNESCO pact to remove tariffs on the importation of educational, scientific, and cultural materials. Among the other ten countries that have ratified it are Cambodia and Laos. Now we have not the slightest idea where Viet-Nam is, nor have we any recollection of ever having heard of the country before. Our ignorance of Laos is almost equally great and we are not at all sure about Cambodia.

Negroes in Engineering

Engineering careers are being increasingly opened to Negroes. RCA-Victor has been scouting for Negro engineering talent for more than three years, has hired 16, nine of them this year, and reports that probably every reasonably qualified Negro engineering graduate has received at least one attractive offer of engineering employment in American industry. Negro engineers are practically if not entirely absent in American mining but there are a few Negro metallurgists and chemists and at least one is an AIME member.

Personals

Frank L. Bader has been appointed to the sales staff of Lehigh Navigation Coal Co. in the Philadelphia territory.

H. C. Burrell, former coordinator of raw materials, Pittsburgh, Penna., has been made manager of raw materials for Columbia-Geneva Div. of U. S. Steel Co., San Francisco.

J. H. Cazier has resigned as general superintendent of Bagdad Copper Corp., Bagdad, Ariz., to devote full time with **E. A. Scholz** to the operation of the Copper King, Phillips and Black Pearl Mines near Bagdad. They have operated the Copper King zinc mine for more than two years. Production started from the Black Pearl and Phillips tungsten mines during the past year.

Robert T. Chapman has resigned from Reynolds Mining Corp. to join Westmoreland Manganese Corp. as general superintendent.

Annan Cook, with Kennecott Copper Corp.'s exploration dept. for the past three years, has been appointed district geologist for the eastern states.

F. J. Cservenyak has succeeded **H. W. St. Clair,** now in charge of the U. S. Bureau of Mines alumina plant at Laramie, Wyoming, as chief of the Light Metals Branch of the Bureau. **D. D. Blue** becomes assistant chief.

William A. Cummens and **James E. Seykora,** civil engineering graduates of the University of Minnesota, are employed by the Oliver Iron Mining Div. of U. S. Steel Co., on the Mesabi iron range. Mr. Cummens is in the Canisteo district and Mr. Seykora is in the Hibbing-Chisholm district.

Robert M. Dreyer, geology department chairman, has been granted a leave of absence from the University of Kansas to be geologist in charge of the uranium div. for Kerr-McGee Oil Industries. **Professor R. C. Moore** will take Professor Dreyer's place.

Henry M. Eickenberry is working for the National Lead Company of Ohio as a technologist.

Julian W. Feiss, formerly assistant to deputy Defense Materials Procurement Administrator, has been appointed staff geologist of the exploration dept. of Kennecott Copper Corp.

E. S. Frohling has been transferred from Western Machinery Co.'s New York office to the Birmingham office.

C. V. O. Hughes has been promoted to the position of assistant manager of Florida Dept. of Virginia-Carolina Chemical Corp. Mr. Hughes joined this organization in May 1951 as a mining engineer.

Robert M. Hurst is employed as a trainee with the American Smelting & Refining Co. at Vanadium, N. Mex.

Leland H. Johnson has been named chief engineer, Ore Mines and Quarries Div., Tennessee Coal & Iron Div., U. S. Steel.

Harold Kirkemo has resigned from the staff of Anaconda Copper Mining company's Spokane exploration office to accept a position with the United States Geological Survey. He will be assigned to the USGS Spokane regional office and will work on the defense minerals exploration program.

Hans Lundberg, recently returned from a visit to Europe, where his company has completed a number of aerial geophysical surveys, notably in Sweden and Portugal.

Robert M. Moyle, assistant district engineer (Mesabi iron range) for the Oliver Iron Mining Div., U. S. Steel Co., has just been assigned to the firm's Duluth general mining offices. **Milton R. Sermon** has been assigned to the position of assistant district engineer at Hibbing to succeed Mr. Moyle.

Elmer Olson, who was resident engineer on the Marquette range for Jones & Laughlin Steel Corp., has been transferred to the Mesabi range as resident engineer for the firm's Minnesota Ore Div. **C. H. Sleeman,** assistant chief engineer of the Minnesota Ore Div. has been appointed general mining engineer at Virginia, Minn.

Hollis G. Peacock has returned to the employ of U. S. Smelting, Refining & Mining Co. Mr. Peacock was first employed by the company at its Fairbanks, Alaska operations in 1936. He joined the company's Utah staff early in 1938 and became the company's resident geologist in the Bingham district in 1945, which position he held until July, 1951, when he resigned to become chief geologist for Chief Consolidated Mining Co.

James A. Rabbitt, consulting engineer, International Nickel Co., and an authority on Japan and the Far East, retired from active service. Mr. Rabbitt, who has been with Inco for more than 22 years, will continue in an advisory capacity on Far Eastern affairs.

Personals, Continued

JOHN C. RUSSELL

John C. Russell resigned his position with the consulting engineering dept. of the Dorr Co., to become chief research metallurgist at the new central laboratories of Rhodesian Anglo American, Ltd., Kitwe, Northern Rhodesia. Russell also held the positions of minerals specialist with the U. S. Government and of metallurgical engineer with the American Cyanamid Co.

J. W. Ribotto was named metallurgical engineer at the Magna concentrator of the Utah Copper Div., Kennecott Copper Corp., replacing **C. G. Williams** who retired.

Harrison Schmitt examined several mining properties in Mexico for the Defense Materials Procurement Agency, Foreign Expansion Div.

Heath Steele, vice-president of the American Metal Co., and president of its subsidiary, Compania Minera de Penoles, S. A., Monterrey, N. L., Mexico, is visiting the company's operations in Mexico. He will also visit the operations of another subsidiary, the Southwest Potash Corp., which are just being started near Carlsbad, N. Mex.

A. G. Wolf, C. O. Stephens and **H. W. Strickland** have been appointed vice presidents of Texas Gulf Sulphur Co. **Mr. Wolf** was placed in charge of offices opened in Houston for the purpose of acquiring new properties. His recommendations led to acquisition of sulphur reserves on several salt domes, including Boling in Wharton County, Texas. Stephens supervised the construction of the largest plant built for extraction of sulphur from sour gas.

Koehler S. Stout will fill the professorship left vacant by the death of Professor O. A. Dingman in the Mining Dept. at the Montana School of Mines.

Roland Tragitt, mining engineer with the U. S. Dept. of the Interior, Bureau of Land Management (formerly the General Land Office) has changed his headquarters from Redlands, Calif., to San Francisco, Calif.

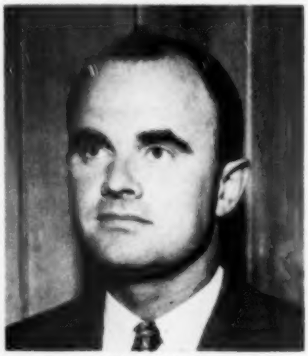
Ernest E. Thurlow was transferred from Spokane, Wash., and is now chief, Salt Lake Exploration Branch, Division of Raw Materials, U. S. Atomic Energy Commission, Salt Lake City, Utah.

Glen L. Waterman has been appointed chief geologist of Chile Exploration Co., Chuquicamata, Chile.

Charles W. Yetter was appointed general manager of all operations of Standard Ore & Alloys Corporation's subsidiary in Mexico, "Corporacion Minera y Beneficiadora Anahuac, S. A.," Mexico, D. F., Mexico.

Wm. Bourne Wood has taken the position of safety engineer with Folley Brothers in sinking the Deep Ruth and Kollinske shafts for the Kennecott Copper Corp. at Ruth, Nev. He has taken the position vacated by **Jack Seerley** who has been promoted to assistant general superintendent on this job.

George R. Schaefer's position has been changed to general manager of Minas de Matahambre, S. A.



T. M. WARE

Thomas M. Ware was recently made vice president in charge of the engineering div. and **E. D. McDougal, Jr.** has become vice president and general counsel of International Minerals & Chemical Corp.

Carlos R. Tapia has resigned from Compania Minera Unificada, Potosi, Bolivia, to accept a position as general mine foreman with Volvan Mines Co. in Lima, Peru.

John T. Shimmin, Jr. is now shift foreman with Magnesium Recovery Dept., Titanium Metals Corp. of America, at Henderson, Nev.

James H. Pierce, chairman of Pierce Management, Inc., visited Mexico, Greece and Spain in connection with several projects.

E. M. LINDENAU

E. M. Lindenau has left Minas San Andres, Honduras, where he was manager of the New Indria Honduras Mining Co. He returned to the Thunderbird Ranch, Tucson, Ariz.

Charles W. Berry, a recent graduate of Lehigh University, and **Robert F. Jaska,** a recent graduate of the University of Minnesota, have joined the mining engineering dept. of the Oliver Iron Mining Div., U. S. Steel Co., on the eastern Mesabi range, Minnesota.

Charles W. Walton has been appointed senior research analyst of the Chemical Construction Corp.

J. R. Gross has been appointed engineer at the Wentworth iron mine by the Minnesota Ore Div. of the Jones & Laughlin Steel Corp. The Wentworth is the most easterly active mine of the firm on the Mesabi iron range.

E. W. Geist, who has been superintendent of the Groveland pilot plant of the M. A. Hanna Co. at Randville, Mich., Menominee range, has returned to his former duties at the research laboratory of the company in Hibbing, Minn., on the Mesabi range.

Jean Lentz has been appointed engineer at the Hill-Annex iron mine, Calumet, Minn., of the Minnesota Ore Div., Jones & Laughlin Steel Corp. He succeeds **R. H. Ledin,** who is now engineer at the Lind-Greenway mine, also on the Mesabi iron range.

Rodney Bleifuss, a recent graduate with a Masters degree in geology from the University of Minnesota, has joined the staff at the Research Laboratory of Oliver Iron Mining Div., U. S. Steel, Duluth, Minn.

Douglas C. Blackwell, metallurgical engineer and **Derral L. Thompson,** mechanical engineer, both recent graduates of the South Dakota School of Mines, are now employed by the Oliver Iron Mining Div. of U. S. Steel Co. on the Mesabi iron range.

W. R. Benedict, (Member 1946) a well known figure in the metallurgical field, died on July 9, 1952. He had been employed by many well known mining companies such as San Luis Mining Co., Basque Mining Co., New Cornellia Mining Co., Cia Minera de San Patricio and Mexican Mines of El Oro. Mr. Benedict had been assayer, metallurgist, flotation shift boss, shift foreman and mill superintendent. Born in Berlin, Conn., his last position was metallurgist with the U. S. Bureau of Mines in Boulder, Nev.

John Edward Clark (Member 1911), who died April, 1952, had a varigated mining career, spread through the operations of several mining companies. He was an assistant to the consulting chemist of Selby Smelting & Refining Co., and later he was a sampler, surveyor assayer, and milling employee for Monica Mines, Kirkland, Ariz. He was also employed on cyaniding and milling work by Golden Star Mining and Milling Co., Polaris, Ariz., and as a chemist and metallurgical engineer for La Barrauca Mines, Mexico. Mr. Clark bequeathed $5000 to the AIME.

H. J. Fraser (Member 1940), vice president in general charge of all plant operations in the United States of the International Nickel Co., died on August 22 after a brief illness. Born in Brockville, Ontario, the son of Oliver K. and Margaret A. Fraser, he attended Queen's University at Kingston, Ontario, graduating in 1923 with a B.S. degree. Mr. Fraser joined the Huntington, W. Va. works of the International Nickel Co. in May 1923 serving in various technical and operating capacities before being promoted to the company's New York office in February, 1935, as assistant manager of the Production Dept. He was elected vice president in March 1947, and in the following June was made vice president in general charge of all plant operations in the United States. Mr. Fraser was a member of the Canadian Institute of Mining & Metallurgy, American Society for Metals, the Mining and Metallurgical Society of America and the Canadian Society of New York. He also was a member of the City Midday Club, New York, the Mining Club, New York and the Larchmont Yacht Club, Larchmont, N. Y.

K. N. Ramstead (Junior Member May, 1952), assistant geologist with Kennecott Copper Corp., died June 2, 1952 in an automobile accident at Perryville, Md. Born in New York City, he was educated at Upsala College, East Orange, N. J., and at the University of Oslo. He was previously employed by Ashland Pattern & Model Works of Brooklyn, N. Y.

Vernon J. Nelson (Member 1949) died May 31, 1952. His last position was with Coeur d'Alene Mining Div., Washington Water Power Co., where he was division manager. Mr. Nelson served in the U. S. Army Signal Corps from 1941 to 1946. Upon his graduation from the University of Idaho, he took a position as topographical mapper in the U. S. Forest Service. He started with the Washington Water Power Co. in 1935. He first started working for the Coeur d'Alene Mining Div. in 1946.

Walter G. Crichton (Member 1944) died during May, 1952 after a career which began with graduation from West Virginia University. Crichton was assistant engineer with Blue Creek Coal & Land Co., and Kanawha & West Virginia Railroad. Twenty-nine years of his life were given over to private civil and mining engineering practice. His birthplace was Philipsburg, Pa., June 30, 1888.

Philip R. Ringulet (Member 1937) died in November 1951. Mr. Ringulet was born in Osceola, Wis. His first job was with the Idaho Continental Mining Co. doing test work and general repairs. From 1926 to 1927 he did mill construction and transportation of equipment for the Northern Peru Mining & Smelting Co. Mr. Ringulet also held positions with Sabinal Reduction Co., South American Copper Co., and the Mazapil Copper Co. and Getchell Mine, Inc., Nevada.

John C. Nicholls (Member 1907) died in retirement after serving as assistant to the president, International Nickel Co., Toronto. Following graduation from the University of California with a B.S. in Mining, Nicholls went to Korea with Oriental Consolidated Mining Co. He left Korea in 1908 after six years as assayer, cyanide plant foreman, and as superintendent of two mining properties. After a trip taking him through Hong Kong, Manila, Australia, New Zealand, and Tasmania, he became head sampler, shift boss and mine captain of one of the more important South African gold mines. He was employed at Copper Cliffs by International Nickel in 1912 as an engineer. He became superintendent of mines in 1913, general superintendent of mining and smelting in 1918, and in 1922 was appointed assistant to the general manager. Mr. Nicholls was a member of the Canadian Institute of Mining and Metallurgy and the South African Institute of Engineers.

P. G. Spilsbury (Member 1906), founder and past president of the Arizona Industrial Congress, died May, 1952 after a brief illness. P. G. Spilsbury also represented Arizona, New Mexico, and Nevada in the Association of American Railways. Eleven years ago, Spilsbury went to Washington, and during much of that time served as consultant to Anaconda Copper Corp., and its principal fabricating subsidiaries, American Brass Co. and Anaconda Wire & Cable Co. Spilsbury graduated from Lehigh University. First employed by the Cananea Consolidated Copper Co., in Mexico, he went on to work on other projects in the U. S., Mexico, and Costa Rica. During this period he worked for the Inspiration Consolidated Copper Co., Arizona as a construction engineer. He was the son of a past president of the AIME.

Herbert A. Megraw (Member 1901) died Nov. 3, 1951. Born in Maryland, Mr. Megraw was educated at Cornell University. Upon graduation from college he was employed as an assayer with the United Mexican Mines Assn. For a number of years he was associated with the Engineering and Mining Journal, first on the editorial staff and then as associate editor. He had experience in gold, silver, and lead mines and was proficient in flotation, gravity, concentration and leaching.

W. A. Odgers (Member 1939) died on Aug. 24, 1951. Mr. Odgers spent most of his career working for the Anglo American Corp. of S. A. He held positions as ventilation officer, shift boss, mine captain, and underground manager. He became mine manager for the company. Mr. Odgers was born and educated in the Union of South Africa.

Appreciation of Daniel B. Johnson

by J. H. Ashley

Daniel B. Johnson, 65, manager at the Naica Unit of The Fresnillo Co. in Mexico, died in St. Mary's hospital, Long Beach, Calif., on Aug. 16, 1952 after a short illness.

He was born in Ulverston, Lancastershire, England, and started mining at an early age, learning from the ground up the hard way. Later he emigrated to America to the copper mines of Utah and Nevada. Mr. Johnson, or "hurry up" as he was known to many of his friends, was well known in Nevada and Arizona mining circles and in Mexico where he had been with the Fresnillo Co. for the last twenty-six years.

He was a great lover of baseball, and did a great deal to popularize and maintain enthusiasm for the sport among the employees at Fresnillo. His greatest hobby was mining, and training of younger men in mining. Many a man in a responsible position today owes his success to Dan's interest and training—the method was sometimes rough, but always to the point.

The mining world and his many friends have lost a well loved leader, and mourn his loss.

Andrew B. Crichton (Member 1901) died on July 9, 1952. During his many years of experience, he has been connected with the Logan Coal Co., American Pipe Mfg. Co., and Glenwood Coal Co. He was at one time president of Johnstown Coal & Coke Co. For quite a few years Mr. Crichton conducted a general engineering business. Educated at Philipsburg and Peale high schools, he took international correspondence school courses at State College, Pa. Mr. Crichton was a member of American Water Works Assn., National Geologists and American Assn. for the Advancement of Science.

Enrique Pablo de Romana (Member 1937) died in 1950 according to word received recently by the Institute. He was born at Arequipa, Peru and attended the University of California. Mr. Romana had been associated with M. Hochschild & Co. as a junior engineer at the Rescate mine in the Dept. of Arequipa doing mapping, sampling and estimating ore reserves. He did square set stope mapping and ore estimates for Cerro de Pasco Corp. From 1936 to 1938 he was superintendent of a lead-silver property at Carumas, Moquegua. Mr. Romana was engaged in mine examination work in southern Peru for M. de Romana and M. Hochschild. He held a B.S. degree in mining engineering.

Appreciation of

Hugh M. Shepard

by T. D. Jones

The Extractive Metallurgy Div., AIME lost one of its most ardent supporters when Hugh M. Shepard died suddenly on May 30.

"Shep" as he was affectionately known to all his associates, assisted John Sullivan and Carleton Long in the organization of the EMD and served as its first secretary.

Mr. Shepard was born in Hamilton, Ont. in 1899 and majored in metallurgical engineering at the University of Toronto. He was employed by the American Smelting & Refining Co. in 1923 and started his career at the Perth Amboy plant as a shift boss in the blast furnace dept. He became successively superintendent of blast furnace operations, superintendent of copper refinery operations, and superintendent of the plant in 1930. He was transferred to the Baltimore plant of Asarco as general superintendent in 1936 and became manager in 1940.

Obituaries Cont'd

Jack W. Clark

An Appreciation by Alan Probert

Jack Clark was well known to the entire engineering fraternity of the Washington area as the amiable and efficient Secretary of the local section of the American Institute of Mining and Metallurgical Engineers during the last few years. At the time of his death, he was on a professional field trip as consultant to the Beryllium Corp. and died of respiratory bulbar paralysis neurotropic virus following a severe attack of malaria in Madagascar. The first word received in Washington came through the Department of State from the American Consul at Tananarive under date of August 8. An earlier message concerning his illness had been received through the same channels.

Jack was educated at Berkeley High School in California and attended the University of California, Berkeley, from 1934 to 1940 following two years at Chasey Junior College at Ontario, Calif. From 1949 until 1951 he was employed in the Minerals Div. of the Bureau of Mines in Washington as a commodity specialist, where his comprehensive knowledge of the rare precious metals minerals was made available to the Government.

His overall experience was largely in the field of geology although he had experienced all the growing pains of a young mining engineer in mines, mills and chemical plants, working as mucker, mill man, chemical analyst, foreman. He obtained general experience with a number of well known American companies including Union Mines Development at Grand Junction, Colo., and the United States Vanadium Corp. at Salt Lake City. Some experience was gained at the Nevada-Massachusetts Co. at Mill City, Nevada and specialized knowledge was developed while in the employ of the Spectrographic Service and Supply Inc., where he was head of the Rare Metals Div. Not the least important connection was a period during which he served as geologist to the Atomic Energy Commission with headquarters at Grand Junction, Colo.

Besides his many friends in the industry, numerous because of his personality qualifications, Jack Clark left a son, Donald Wallace Clark and a mother, Mrs. Muriel Clark of Pomona, Calif.

D. R. Schooler (Member 1943) of the mining faculty at Missouri School of Mines died May 30, 1952. Born in Columbus, Mo., he was educated at the Missouri School of Mines where he received a B.S. in mining. Professor Schooler spent 6 years with Centralia Coal Co., Centralia, Ill., as a mining engineer and from 1935 to 1942 was mine superintendent for this company. Professor Schooler had been with the Missouri School of Mines since 1942 where he was an assistant professor of engineering drawing.

John H. Hastings (Member 1915), formerly chief chemist for American Zinc & Chemical Co., Pennsylvania, died. Prior to becoming a chemist he was in turn assistant on mine examination with J. B. Hastings, in Mexico, a miner for Calumet and Arizona Mfg. Co., Arizona, and a sampler for Western Chemical Mfg. Co., Denver.

NECROLOGY

Date Elected	Name	Date of Death
1947	Reginald K. Bailey	Dec. 16, 1951
1935	J. G. Bently	Unknown
1920	Harold E. Boyd	June 24, 1952
1912	Thomas H. Clagett	July 23, 1952
1945	Jack W. Clark	Aug. 8, 1952
1933	W. T. Griffiths	July 30, 1952
1947	Fred B. Heath	July 25, 1952
1942	B. W. Jacobs	Unknown
1944	H. A. McAllister	Unknown
1929	George L. Richert	May 26, 1952
1885	H. Ries	April 1951

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Aug. 31, 1952 was 18,125; in addition 1,555 Student Associates were enrolled.

ADMISSIONS COMMITTEE

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The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Arizona
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Denver—Drinkwater, Donald J. (R. C/S—S-M)
Denver—Shaw, Richard H. (R. C/S—S-M)
Gilman—Phillips, John W. (J)
Golden—Bunge, Fred H. (C/S—A-M)
Grand Junction—Pursley, Richard J. (J)
Grand Junction—Redmond, Robert L. (J)
Ouray—Brechtel, Charles E. (R. C/S—S-J)
Pueblo—Kalish, Philip (J)

District of Columbia
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Pittsburg—Spencer, William C. (R. C/S—S-J)

Massachusetts
Williamstown—Walker, Bryant D. (J)

Michigan
Berkley—Mann, John W. (A)
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Ishpeming—Thompson, Charles D. (R. C/S—S-J)

Minnesota
Minneapolis—Kurtze, Harry A. (M)

Missouri
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Cooke City—McGrath, Thomas G. (M)

New Jersey
Morristown—Cutler, Ralph H., Jr. (A)

New Mexico
Hanover—Snell, Clarence C. (M)
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New York—Schwalbe, Erik A. (M)

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Tennessee
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Houston—Ogg, James H. (C/S—S-J)

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East Africa
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Coming Events

Oct. 3, AIME, National Open Hearth, Southern Ohio Section, Deshler-Wallick Hotel, Columbus, Ohio.

Oct. 10, AIME, Eastern Section, National Open Hearth Steel Committee, Harwich Hotel, Philadelphia.

Oct. 15-17, Canadian Institute of Mining and Metallurgy, annual western meeting, Royal Alexandra Hotel, Winnipeg.

Oct. 20-22, AIME, Institute of Metals Div., fall meeting with National Metal Congress, Hotel Adelphia, Philadelphia.

Oct. 20-24, American Society for Metals, meeting with National Metal Congress, Benjamin Franklin Hotel, Philadelphia.

Oct. 20-24, 40th National Safety Congress, Conrad Hilton, Congress, LaSalle, Morrison and Sheraton Hotels, Chicago.

Oct. 24, Fall Meeting of Mining Branch of the Southern California Section, AIME, Statler Hotel, Los Angeles.

Oct. 24, Illinois Mining Institute, annual meeting, Hotel Abraham Lincoln, Springfield, Ill.

Oct. 28, Assn. of Consulting Chemists and Chemical Engineers, Inc., annual symposium, Hotel Belmont Plaza, New York.

Oct. 30-31, AIME, Fuels Conference, Coal Div.; **ASME,** Fuels Div., Bellevue-Stratford, Philadelphia.

Nov. 4, AIME, Morenci Sub-section, Longfellow Inn, Morenci, Ariz.

Nov. 6-8, New Mexico Mining Assn. and **International Mining Days,** joint convention, Alvarado Hotel, El Paso.

Nov. 7-8, Birmingham Technical Meeting of Southeast Section, AIME, Birmingham.

Nov. 12, AIME, Connecticut Section, Hartford, Conn.

Nov. 18, AIME, Buffalo Section, National Open Hearth Steel Committee, Hotel Statler, Buffalo.

Nov. 19, American Mining Congress Coal Div. Conference, Wm. Penn Hotel, Pittsburgh.

Nov. 20-21, American Society for Quality Control, mid-west conference, Claypool Hotel, Indianapolis.

Dec. 2, American Mining Congress, annual membership meeting, University Club, New York.

Dec. 4-6, AIME, Electric Steel Furnace Conference, Hotel William Penn, Pittsburgh.

Dec. 5, Atlanta Meeting of Southeast Section, AIME, Ansley Hotel, Atlanta.

Dec. 7-10, American Institute of Chemical Engineers, annual meeting, Hotels Cleveland and Carter, Cleveland.

Dec. 8, AIME, Arizona Section, all-day meeting, Tucson.

Jan. 14, 1953, AIME Connecticut Section, Hammond Metallurgical Laboratory, Yale University, New Haven, Conn.

Feb. 16-19, AIME, annual meeting, Statler Hotel, Los Angeles.

Mar. 16-20, National Assn. of Corrosion Engineers, annual conference and exhibition, Hotel Sherman, Chicago.

Mar. 23-27, ASM western metal congress and exposition, Pan-Pacific Auditorium, Los Angeles.

Apr. 12-May 23, Empire Mining and Metallurgical Congress, Australia-New Zealand.

Apr. 20-22, AIME, National Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Hotel Statler, Buffalo.

May 14-16, Pacific-Northwest Metals and Minerals Conference of 1953, joint meeting of Metals Branch and Industrial Minerals Div., Ben Franklin Hotel, Seattle.

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